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ACCIDENT EXPERIENCE OF THE COAL MINES OF UTAH FOR THE PERIOD 1918 TO 1929[1]

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From 1870 to the end of 1929 the coal production of Utah has been somewhat less than 110,000,000 tons. During this period at least three major disasters have occurred with fatalities totaling about 380, establishing a decidedly bad record of approximately 3.5 persons killed by major mine explosions per million tons of coal produced. Even during the period when there have been no disasters, such as the years 1914 to 1920, inclusive, the fatalities in ordinary working were about 5.4 per million tons produced, against 3.7 per million tons for the whole United States for the same years. The 5-year period following the World War, 1918 to 1922, inclusive, had 4.7 fatalities per million tons; hence, the fatality record in the coal mines of Utah has uniformly been in excess of the average of the coal mines of the entire country.

Notwithstanding this poor record, Utah has given to the coal-mining world many excellent safety methods and practices, and Utah coal mines in general take greater measures toward safe mining than do the coal mines of any other State; if practices and methods allowed and pursued in coal mines of other States were tried in Utah coal mines, there would be a succession of disasters.

OPERATING CONDITIONS

Utah's principal producing coal mines, operated by about 23 companies and numbering 42 in 1929, have comparatively thick beds. Little coal under 5 feet thick is being worked and practically none under 4 feet; much, if not most, of the coal comes from beds in excess of 8 feet, some running as high as 30 feet. In thick beds in which coal is left on floor and roof, any dust found or made is entirely combustible. In addition, the thick beds induce the use of large-capacity cars, the handling of which in rooms and other workings on fairly heavy grades (up to 10 or 12 per cent) tends to cause haulage accidents. The thick beds give high ribs and roof, in which it is difficult to prevent slabs of coal or rock from dropping on workers and also

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difficult to keep free of gas and coal-dust. The coal has high volatile content, generally over 40 per cent and in some instances nearly to 50 per cent; hence it is highly inflammable. Most of the coals have large resin content, which is usually in dust form and hence is almost as inflammable as black blasting powder. At least one-third of the mines give off methane and several give off other inflammable gas with a gasoline or kerosene odor. Several mines have oil seepages in the roof or floor; in some instances, there is enough oil to necessitate frequent baling into barrels and removal from the mine. Some mines have a difficult safety, as well as efficiency, problem in the mining of two or more beds with only a few feet of rock between them, the rock thickness varying from place to place. Most of the mines are working under heavy cover, which approximates 3,000 feet in some instances and presents decidedly difficult problems in controlling roof. Probably the worst handicap, however, is the dry climate; this enforces the use of surface air, which in all seasons of the year abstracts moisture from the mine workings. This makes the coal-dust the more inflammable and robs Utah coal mines of the definite advantage obtained in other localities where "sweating" takes place during several months of the year and tends to make the mine dust noninflammable.

A study has been made of various conditions and results in connection with safety in Utah's coal mines for the years 1918 to 1929, inclusive. Table 1, following, compiled from United States Bureau of Mines publications, gives interesting data on production and fatalities:

Table 1.- Coal production, men employed, and men killed in Utah mines, 1918-1929

Year	Coal, short tons	Number employed	Fatal accidents	Annual coal production per employee, tons	
				Utah	Bituminous mines of the United States
1918	5,136,825	4,160	19	1,235	942
1919	4,631,323	3,857	27	1,201	749
1920	6,005,199	4,504	35	1,333	889
1921	4,078,784	4,422	15	922	627
1922	4,992,008	4,721	22	1,057	614
1923	4,720,217	4,381	23	1,077	801
1924	4,488,157	4,330	200*	1,037	781
1925	4,690,342	4,441	24	1,056	884
1926	4,373,793	3,545	21	1,234	966
1927	4,781,480	3,339	26	1,432	872
1928	4,842,544	3,352	18	1,445	959
1929	5,072,527	3,458	34	1,492	1,064

* Castlegate disaster caused 171 deaths.

COAL PRODUCTION AND FATALITIES

The number of mines producing coal in Utah has varied only slightly from 32 in 1918 to 42 in 1929. Many of these are small wagon mines serving communities which are considerable distances from railroads. Most of the coal produced in Utah comes from mines operated by a few relatively large companies. Producers mining in excess of 10,000 tons per year have steadily grown from 13 in 1918 to 22 in 1929. Of these, 8 produced in excess of 100,000 tons in 1918, whereas in 1929, 14 produced in excess of 100,000 tons.

The number of men employed in the mining of coal in Utah has varied somewhat with the production.

The fatal accidents occurring in the coal mines of Utah during the period 1918 to 1929 have totaled 464; the yearly average for the 12 years is 38.66 fatalities; if the major disasters of 1924 at Castlegate with 171 deaths and at Rains with 5 are excluded, the yearly average is 24 fatalities.

The production of coal per man employed in Utah is comparatively high, and for the years studied, has been equalled by other States only on four occasions. During 1918, Wyoming exceeded the production record per employee of Utah by 14 tons. In 1925, both Montana and West Virginia recorded an employee-production figure greater than Utah, respectively, by 80 tons and 55 tons. Again in 1928, Montana exceeded Utah by 72 tons. Table 2 lists the fatal accidents during the period 1918-1929 by causes.

During the 12-year period the greatest number of deaths, 182, or 39.2 per cent, were due to the explosion of gas or coal-dust. As the Castlegate disaster in 1924 accounted for 171 fatalities, the excessive number of deaths due to this cause can not be taken as an index of normal fatalities from gas and coal-dust explosions in Utah, though natural conditions in Utah's coal mines are so extremely dangerous that widespread disasters similar to that at Castlegate in 1924 are likely to occur at any time upon relaxation of safety precautions.

Falls of coal or rock was the cause of the next largest number of fatalities, 163, or 35.1 per cent. Death from falls of coal or rock, although varying in numbers year by year, was the principal cause of fatalities in each of the 12 years, except in 1924, the year of the Castlegate and Rains disasters. Had proper supervision been exercised, adequate timber provided and required to be set in place, and chance-taking penalized, many of these deaths might have been avoided.

Deaths from mine cars and locomotives for the period were 61, or 13.1 per cent of the total; the regularity of their occurrence year after year indicates a failure to apply any effective effort to correct the conditions causing them. However, haulage conditions are such that unless definite precautionary measures are at all times kept in effect against haulage accidents, there are likely to be numerous mishaps and they are likely to be severe.

Table 2. - Fatal accidents in the coal mines of Utah by causes, 1918-1929

	1918	1919	1920	1921	1922	1923	1924	1925	1926	1927	1928	1929	Total
Underground:													
Falls of coal or rock	7	15	18	5	11	11	12	16	15	18	11	24	163
Mine cars and locomotives	4	4	10	5	6	6	7	4	4	3	3	5	61
Explosions of gas or coal-dust	-	1	-	-	-	2	178	-	-	-	-	1	182
Explosives	1	1	-	-	2	1	-	-	-	1	-	-	6
Suffocation from mine gases	-	-	1	-	-	-	1	-	-	-	-	-	2
Electricity	-	-	-	1	1	-	-	1	1	1	-	1	6
Animals	-	-	-	-	-	1	-	-	-	-	-	-	1
Mining machines	1	3	1	2	-	-	-	-	-	-	1	-	8
Other causes	1	-	-	-	-	-	-	-	1	-	1	1	4
Total underground	14	24	30	13	20	21	198	21	21	23	16	32	433
Surface:													
Mine cars and locomotives.	1	1	1	-	1	-	2	-	-	-	2	-	8
Electricity	1	-	3	1	1	-	-	-	-	-	-	1	7
Railway cars and locomotives	2	1	-	1	-	-	-	-	-	-	-	-	4
Other causes	1	1	1	-	-	2	-	3	-	3	-	1	12
Total surface	5	3	5	2	2	2	2	3	-	3	2	2	31
Grand total	19	27	35	15	22	23	200	24	21	26	18	34	464

COMPARISON OF UTAH'S FATALITY RATES WITH THOSE OF THE BITUMINOUS MINES OF THE UNITED STATES

The fatality rate for the coal mines of Utah as compared with all the bituminous mines of the United States, both on the basis of number of employees working and the tonnage produced, can hardly be considered favorable, as may be seen from Table 3.

Table 3. - Fatal accidents in the coal mines of Utah compared with all bituminous mines in the United States, 1918 - 1929

Year	Fatality rate per thousand 300-day workers		Fatality rate per million tons mined		Coal produced per fatality, tons	
	Utah	United States	Utah	United States	Utah	United States
1918	5.28	3.97	3.70	3.50	270,360	285,552
1919	8.77	4.16	5.83	3.62	171,530	275,983
1920	9.26	3.79	5.83	3.13	171,577	319,296
1921	6.72	4.38	3.68	3.48	271,919	287,239
1922	6.86	5.16	4.41	3.99	226,909	250,753
1923	9.88	4.65	4.87	3.46	205,227	289,076
1924	75.81	5.39	44.34	3.94	22,554	253,770
1925	9.05	4.79	5.12	3.53	195,431	283,562
1926	9.12	4.86	4.57	3.60	218,690	277,660
1927	11.16	4.60	5.44	3.36	183,903	297,224
1928	8.44	4.90	3.72	3.45	254,870	289,447
1929	14.01	4.63	6.59	3.19	149,191	312,964
Average	14.18	4.56	7.98	3.51	124,736	285,001

COMPARISON OF FATALITY RATES FOR INDIVIDUAL STATES WITH RATES FOR THE UNITED STATES

The fatality rates for the coal mines of the United States and also comparative figures for each State are shown in Tables 4, 5, and 6 covering 5-year periods, 1919 to 1923, 1920 to 1924, and 1925 to 1929. The figures are shown separately for each of the main causes of fatal accidents--namely, falls of roof and coal, haulage, gas and dust explosions, and miscellaneous. The fatality rates from each of these causes as shown for the United States represents the number of deaths from that cause for each million man-hours of exposure of all underground employees. The figures for the several States indicate the percentage which the State rates bear to the corresponding rate for the United States. The States are listed in the order of their standing as compared with the fatality rates for the United States, and have been reproduced from United States Bureau of Mines Bulletin 283, "Coal-mine accidents in the United States, 1926"; and Bulletin 341, "Coal-mine Accidents in the United States, 1929" by W. W. Adams, published in 1927 and 1931, respectively.

[illegible]

[illegible]

[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

[illegible]

[illegible]

TABLE 4.—*Index numbers, comparing coal-mine fatality rates per million man-hours, underground, for different States, with the average rate for the United States, 1919 to 1923 (revised)*

Falls of roof and coal

State	Index number
Alaska, California, Idaho, Nevada, and Oregon	------
South Dakota	------
Texas	13
Georgia and North Carolina	60
Michigan	61
North Dakota	65
Illinois	77
Pennsylvania (bituminous)	[illegible]
Missouri	[illegible]
Indiana	[illegible]
Oklahoma	[illegible]
Maryland	[illegible]
Pennsylvania (anthracite)	89
Iowa	[illegible]
Tennessee	[illegible]
Kansas	[illegible]
Alabama	96
Kentucky	99
Washington	99
United States (1.010)	100
Virginia	104
Montana	133
Ohio	138
Arkansas	148
West Virginia	164
Wyoming	178
Colorado	186
New Mexico	197
Utah	230

Haulage (underground)

State	Index number
Arkansas	------
Alaska, California, Idaho, Nevada, and Oregon	------
Georgia and North Carolina	------
South Dakota	------
Missouri	18
Kansas	23
Maryland	33
Texas	35
Iowa	39
Pennsylvania (bituminous)	76
Pennsylvania (anthracite)	85
Oklahoma	87
Michigan	87
Virginia	88
Tennessee	89
Alabama	90
United States (0.879)	100
Ohio	100
Indiana	109
Kentucky	110
North Dakota	111
Illinois	119
Washington	123
Montana	125
Wyoming	152
New Mexico	153
West Virginia	170
Colorado	178
Utah	281

Gas and dust explosions

State	Index number
Virginia	------
Maryland	------
Texas	------
Michigan	------
Georgia and North Carolina	------
South Dakota	------
Missouri	10
Iowa	11
Ohio	14
Montana	18
Tennessee	25
Utah	49
Pennsylvania (bituminous)	51
West Virginia	54
Kentucky	63
Illinois	64
Pennsylvania (anthracite)	70
Washington	86
United States (0.238)	100
North Dakota	120
Indiana	153
Kansas	192
Alaska, California, Idaho, Nevada, and Oregon	235
Arkansas	255
Alabama	353
Colorado	354
Okalhoma	456
Wyoming	904
New Mexico	1,931

Miscellaneous (underground)

State	Index number
Alaska, California, Idaho, Nevada, and Oregon	------
Georgia and North Carolina	------
South Dakota	------
Maryland	22
Tennessee	43
Iowa	59
Pennsylvania (bituminous)	67
Missouri	69
Virginia	69
Illinois	71
Michigan	71
Kansas	86
Alabama	87
West Virginia	96
Kentucky	96
United States (0.864)	100
Ohio	106
Texas	113
Colorado	116
Wyoming	116
Oklahoma	123
New Mexico	137
Montana	139
Indiana	149
North Dakota	150
Utah	153
Pennsylvania (anthracite)	163
Arkansas	196
Washington	205

Total underground (including shaft)

State	Index number
South Dakota	----
Alaska, California, Idaho, Nevada, and Oregon	20
Georgia and North Carolina	31
Texas	32
Maryland	54
Missouri	60
Michigan	61
Iowa	66
Pennsylvania (bituminous)	73
Tennessee	74
Illinois	81
Virginia	83
Kansas	90
North Dakota	95
Kentucky	97
United States (2.011)	100
Pennsylvania (anthracite)	101
Indiana	108
Ohio	111
Montana	119
Washington	121
Alabama	124
West Virginia	135
Oklahoma	140
Arkansas	143
Colorado	191
Utah	200
Wyoming	244
New Mexico	376

TABLE 5.—*Index numbers, comparing coal-mine fatality rates per million man-hours, underground, for different States, with the average rate for the United States, 1920 to 1924 (revised)*

Falls of roof and coal

State	Index number
Alaska, California, Idaho, Nevada, and Oregon	------
South Dakota	------
Texas	15
Michigan	58
North Dakota	66
Illinois	75
Tennessee	77
Oklahoma	78
Pennsylvania (bituminous)	81
Indiana	83
Kansas	85
Pennsylvania (anthracite)	86
Missouri	91
Iowa	93
Alabama	95
United States (1.047)	100
Maryland	107
Kentucky	112
Virginia	113
Georgia and North Carolina	115
Washington	130
Ohio	139
Arkansas	148
West Virginia	155
Montana	167
Colorado	172
Wyoming	186
New Mexico	198
Utah	219

Haulage (underground)

State	Index number
Arkansas	------
Alaska, California, Idaho, Nevada, and Oregon	------
Georgia and North Carolina	------
South Dakota	------
Missouri	20
Kansas	24
Texas	29
Maryland	35
Iowa	40
Pennsylvania (bituminous)	76
North Dakota	76
Pennsylvania (anthracite)	79
Oklahoma	79
Virginia	80
Tennessee	88
Alabama	90
Michigan	92
United States (0.867)	100
Ohio	103
Indiana	103
Illinois	111
Kentucky	114
Washington	131
New Mexico	132
Montana	135
Wyoming	169
West Virginia	182
Colorado	187
Utah	206

Gas or dust explosions

State	Index number
Virginia	------
Missouri	------
Maryland	------
Montana	------
Texas	------
Michigan	------
Georgia and North Carolina	------
South Dakota	------
Iowa	4
Ohio	8
Tennessee	36
Kentucky	50
Pennsylvania (bituminous)	51
Pennsylvania (anthracite)	62
Illinois	66
Indiana	86
North Dakota	94
United States ([illegible].297)	100
West Virginia	118
Kansas	134
Washington	149
Colorado	158
Alaska, California, Idaho, Nevada, and Oregon	180
Arkansas	204
Alabama	232
Oklahoma	275
Wyoming	968
New Mexico	1,537
Utah	2,273

Miscellaneous (underground)

State	Index number
Alaska, California, Idaho, Nevada, and Oregon	------
Georgia and North Carolina	------
South Dakota	------
Maryland	28
Iowa	54
Missouri	58
Kansas	59
Pennsylvania (bituminous)	69
Illinois	75
Virginia	77
Tennessee	82
Ohio	82
Texas	95
Michigan	99
United States (0.838)	100
Alabama	102
Kentucky	106
West Virginia	107
Wyoming	110
Colorado	112
Utah	144
Pennsylvania (anthracite)	145
Indiana	149
Montana	150
New Mexico	164
North Dakota	165
Oklahoma	187
Washington	203
Arkansas	230

Total underground (including shaft)

State	Index number
South Dakota	------
Alaska, California, Idaho, Nevada, and Oregon	26
Texas	39
Georgia and North Carolina	59
Missouri	60
Michigan	62
Iowa	64
Maryland	65
Pennsylvania (bituminous)	73
Tennessee	74
Kansas	77
Illinois	80
Virginia	85
North Dakota	88
Pennsylvania (anthracite)	91
Indiana	99
United States (2.040)	100
Kentucky	102
Ohio	105
Alabama	118
Oklahoma	125
Montana	136
Washington	143
Arkansas	145
West Virginia	140
Colorado	163
Wyoming	234
New Mexico	375
Utah	518

Table 6.—*Index numbers, comparing coal-mine fatality rates per million man-hours, underground, for different States, with the average rate for the United States, 1925 to 1929*

Falls of roof and coal		Haulage (underground)		Gas or dust explosions		Miscellaneous (underground)		Total underground (including shaft)	
State	Index No.	State	Index No.	State	Index No.	State	Index No.	State	Index No.
Georgia and North Carolina	......	Alaska, California, Idaho, Nevada, Arizona, and Oregon	......	Missouri	......	South Dakota	......	South Dakota	......
South Dakota	......	South Dakota	......	Maryland	......	Maryland	34	Texas	45
Michigan	53	Texas	23	New Mexico	......	Pennsylvania (bituminous)	56	Maryland	57
Texas	54	Kansas	30	Montana	......	Virginia	68	Michigan	61
Maryland	64	Missouri	44	Texas	......	Wyoming	70	Missouri	68
North Dakota	69	Iowa	57	Michigan	......	Illinois	74	Illinois	73
Illinois	71	Pennsylvania (anthracite)	68	North Dakota	......	Kentucky	75	Kansas	73
Missouri	71	Tennessee	68	Alaska, California, Idaho, Nevada, Arizona, and Oregon	......	New Mexico	75	Alaska, California, Idaho, Nevada, Arizona, and Oregon	77
Alaska, California, Idaho, Nevada, Arizona, and Oregon	73	Pennsylvania (bituminous)	78	South Dakota	......	Tennessee	77	North Dakota	80
Kansas	74	Oklahoma	78	Ohio	6	Texas	84	Pennsylvania (bituminous)	81
Alabama	80	Illinois	88	Virginia	12	Iowa	95	Pennsylvania (anthracite)	91
Indiana	82	Ohio	89	Utah	14	Utah	96	Kentucky	92
Pennsylvania (bituminous)	83	Indiana	91	Iowa	25	Michigan	96	Tennessee	98
Tennessee	85	United States (0.379)	100	Wyoming	35	United States (0.312)	100	Ohio	99
Pennsylvania (anthracite)	92	Maryland	104	Illinois	59	West Virginia	101	United States (2.125)	100
Iowa	92	Kentucky	106	Pennsylvania (anthracite)	62	Alabama	120	Virginia	101
Oklahoma	92	Arkansas	106	Kentucky	62	Ohio	129	Alabama	103
United States (1:[illegible])	100	Michigan	106	Kansas	64	Indiana	129	Montana	103
Kentucky	101	Virginia	107	Colorado	70	Kansas	129	Indiana	123
Arkansas	103	Alabama	109	West Virginia	96	Colorado	136	Iowa	123
Montana	107	North Dakota	122	Washington	98	Pennsylvania (anthracite)	144	West Virginia	128
Ohio	119	Montana	130	United States (0.312)	100	North Dakota	148	New Mexico	131
West Virginia	131	Colorado	134	Pennsylvania (bituminous)	105	Missouri	150	Colorado	136
Virginia	133	Washington	151	Alabama	163	Montana	158	Wyoming	136
Colorado	155	Wyoming	159	Tennessee	206	Arkansas	240	Washington	174
Washington	157	West Virginia	165	Indiana	303	Oklahoma	246	Arkansas	174
New Mexico	161	Utah	170	Arkansas	443	Alaska, California, Idaho, Nevado, Arizona, and Oregon	263	Utah	227
Wyoming	174	New Mexico	197	Oklahoma	1,864	Washington	338	Oklahoma	372
Utah	341	Georgia and North Carolina	986	Georgia and North Carolina	10,978	Georgia and North Carolina	599	Georgia and North Carolina	1,876

The index tabulation (Table 4) covering the 5-year period 1919 to 1923, inclusive, hence before the Castlegate and Rains disasters, shows that Utah had 2.2 as many fatalities on an exposure basis from falls of roof and coal as the average for the coal mines of the United States, and had by far the worst record of the coal-mining States in fatality rate from falls of roof and coal. Utah had also much the poorest record of the coal-mining States in fatality rate from haulage its rate being 2.81 times that of the industry throughout the United States. For the 5-year period 1919-1923, Utah's fatality rate from mine explosions was good, being less than half of the mine-explosion fatality rate for the whole country. For this 5-year period before the Rains and Castlegate disasters occurred, Utah's coal mines had just double the total underground fatality rate for the industry throughout the country, and only Wyoming and New Mexico had poorer rates.

The tabulation (Table 5) for the exposure fatality rates of the various coal-producing States for the 5-year period 1920-1924 shows Utah's coal mines in a decidedly dark light, because for this period it has the worst rate of all the States from falls of roof and coal, from haulage, from gas or dust explosions, and from all underground fatalities; this dark record for the period 1920-1924 is due in part to the Castlegate and Rains disasters in 1924; but these disasters can not be held responsible for the fact that from 1920 to 1924, inclusive, the exposure fatality rate of Utah's coal mines was 2.19 times that of the mines of the whole country from falls of roof and coal and 2.96 times that from haulage.

And the index tabulation (Table 6) for the 5-year period 1925 to 1929, inclusive, which does not include the deaths from the Castlegate and Rains disasters, again shows Utah's coal mines in a bad light, indicating that as yet Safety in Utah's coal mines has not been brought much if any nearer to the average of the safety of the coal mines of the United States.

COMPENSATION AND MEDICAL COST OF ACCIDENTS IN UTAH

The cost of accidents in the mining of coal in Utah, whether considered on the basis of compensation premiums paid or of the compensation and medical costs awarded, has been no small item in contributing to the expense of production. As costs of coal-mine accidents are available only by fiscal years, the following tabulations (Tables 7, 8, 9, 10, 11) for the fiscal years 1918-1919 to 1928-1929, taken from the reports of the Industrial Commission of Utah, give details relative to the several types of accidents, their rates per 1,000 employees, and the actual costs resulting from them.

Table 7. - Fatal accidents in the coal mines of Utah for the fiscal years 1918-1919 to 1928-1929

Fiscal year	Fatal accidents	Rates per 1,000 employees	Compensation and medical costs
1918-1919	24	6.8	$ 51,304
1919-1920	30	6.9	58,810
1920-1921	31	6.3	72,394
1921-1922	19	3.9	56,816
1922-1923	24	4.7	67,934
1923-1924	190	35.4	688,849
1924-1925	33	7.0	146,659
1925-1926	19	3.9	71,823
1926-1927	28	7.3	101,096
1927-1928	17	4.4	57,788
1928-1929	25	5.9	99,894
Total	440	8.9	$1,473,367
Average per year.	40	8.9	$ 133,942

Table 8. - Accidents causing permanent total disabilities in the coal mines of Utah for the fiscal years 1918-1919 to 1928-1929

Fiscal year	Permanent total disabilities	Rate per 1,000 employees	Compensation and medical costs
1918-1919	-	-	-
1919-1920	-	-	-
1920-1921	2	0.4	$ 24,000
1921-1922	-	-	-
1922-1923	-	-	-
1923-1924	-	-	-
1924-1925	-	-	-
1925-1926	-	-	-
1926-1927	-	-	-
1927-1928	3	0.7	$ 45,000
1928-1929	-	-	-
Total	5	0.1	$ 69,000
Average per year.	0.4	0.09	$ 6,273

Table 9. - Accidents causing permanent partial disabilities in the coal mines of Utah for the fiscal years 1918-1919 to 1928-1929

Fiscal year	Permanent partial disabilities	Rate per 1,000 employees	Compensation and medical costs
1918-1919	19	5.4	$ 21,661
1919-1920	16	3.7	16,616
1920-1921	29	5.9	46,572
1921-1922	23	4.8	17,845
1922-1923	26	5.1	21,512
1923-1924	19	3.5	15,182
1924-1925	35	7.4	40,044
1925-1926	25	5.2	20,098
1926-1927	62	16.3	85,240
1927-1928	50	13.0	53,859
1928-1929	89	21.1	123,859
Total	393	7.9	$ 462,488
Average per year.	35.7	7.9	$ 42,044

Table 10. - Accidents causing temporary injuries in the coal mines of Utah for the fiscal years 1918-1919 to 1928-1929

Fiscal year	Temporary injuries	Rate per 1,000 employees	Compensation and medical costs
1918-1919	1,599	458.6	$ 41,591
1919-1920	2,134	493.9	64,295
1920-1921	2,075	427.6	66,209
1921-1922	1,770	369.5	78,122
1922-1923	1,611	320.8	60,722
1923-1924	1,714	320.1	60,938
1924-1925	1,722	365.8	66,165
1925-1926	1,698	353.8	63,170
1926-1927	1,637	431.6	46,309
1927-1928	1,645	429.7	45,250
1928-1929	1,887	448.5	52,264
Total	19,492	396.5	$ 645,035
Average per year	1,772	396.5	$ 58,640

Table 11. - Total compensible injuries in the coal mines of Utah for the fiscal years 1918-1919 to 1928-1929

Fiscal year	All compensible injuries	Rate per 1,000 employees	Compensation and medical costs
1918-1919	1,642	471.0	$ 114,556
1919-1920	2,180	504.6	139,721
1920-1921	2,137	440.4	209,174
1921-1922	1,812	378.2	152,783
1922-1923	1,661	330.8	150,168
1923-1924	1,923	359.2	764,969
1924-1925	1,790	380.2	252,868
1925-1926	1,742	363.0	155,091
1926-1927	1,727	455.4	232,645
1927-1928	1,715	448.0	201,897
1928-1929	2,001	475.6	276,017
Total	20,330	413.5	$2,649,889
Average per year.	1,848.1	413.5	$ 240,899

The fatal accidents which occurred in the coal mines of Utah for the 11-year period totaled 440, or an average of 40 per year. The fatal accident rates show no marked fluctuation except for the fiscal year 1923-1924 when the Castlegate disaster occurred.

Compared with similar fatal accident rates for all bituminous mines of the United States, the Utah rates are year by year consistently higher, and for most years more than double. For the entire period the fatal accident rate for the coal mines of Utah per 1,000 employees is 8.9 compared with a rate of 2.9 per 1,000 employees for all the bituminous coal mines of the United States for the same period.

There have been few permanent total disabilities in the coal mines of Utah; only 5 cases were recorded in the 11 years.

Permanent partial disabilities during the period totaled 393, giving a yearly average of 35.7 and a yearly rate per 1,000 employees of 7.9. During the last three fiscal years studied there was a marked rise in injuries resulting in permanent partial disabilities, the rates per 1,000 employees increasing from 5.2 in 1925-1926 to 16.3 in 1926-1927, 13.0 in 1927-1928, and 21.1 in 1928-1929, or more than 400 per cent increase in 1928-1929 as compared with 1925-1926. There is good reason to believe that much, if not all of this increase, is due to mechanized loading and other mechanization of mining, which were being rapidly expanded in Utah's coal mines during these three years.

The total temporary injuries for the 11 years amounted to 19,492, with a yearly average of 1,772 and a yearly rate per 1,000 employees of 396.5. The

rates per 1,000 employees for temporary disabilities also increased from 353.8 in 1925-1926 to 431.6 in 1926-1927, 429.7 in 1927-1928, and 448.5 in 1928-1929, or 26 per cent increase in 1928-1929 as compared with 1925-1926. Here again mechanization undoubtedly contributed largely, if not wholly, to the increased injury rate.

Judging from the accident experience shown in the foregoing tabulations for the 11 fiscal years, one can view the mining of coal in Utah only as a very hazardous occupation. The rates indicate that for every thousand men employed, at least 413, or 41.3 per cent, receive a disabling injury each year. Of these 413 injuries about 9 result fatally, about 8 are permanent partial disabilities to be carried through life, and the remaining 396 are temporary disabilities with loss of 4 to 28 or more days.

RELATION OF ACCIDENT CHARGES TO MINING COSTS

The economic relation of accidents to coal-mining costs in Utah is becoming more and more apparent. With the decreasing value of coal at the mines and the increasing cost of compensation insurance, due chiefly to greater numbers of accidents, the small margin of possible profit in coal mining is being definitely decreased in most cases, and in some instances the possible profit is actually turned into a loss by the cost of accidents. Should the tendency to yearly increase of coal-mining accidents continue, the resultant rates for compensation insurance will necessarily increase, possibly to such an extent that they are liable to reduce returns on coal mining to a point where little or no profit may be shown.

During the 11 fiscal years included in this study the compensation and medical costs of accidents have been as follows:

Fatal accidents	$1,473,367
Permanent total disabilities	69,000
Permanent partial disabilities	462,487
Temporary disabilities	645,035
Total	$2,649,889

The yearly average of these costs amounting to $240,899, nearly one-quarter of a million dollars, is most certainly an unduly large sum to be expended for accidents in the mining of an average of 4,794,606 tons of coal per year.

Although the foregoing figures represent the medical costs and compensation awards made as the result of accidents in the coal mines of Utah, they fall far short of the real cost of those accidents. The compensation insurance paid as coverage for accident liability represents more nearly the true costs, as it is the actual direct outlay.

During the 11 years the compensation premiums paid by Utah's coal mines have amounted to $4,125,381, or an average of $375,035 yearly. For the full 11-year period the compensation premium cost has averaged 7.8 cents per ton of coal mined, or 2.8 per cent of the average value of the coal over the tipple.

The compensation rates during the early part of the period were found to be high, $7.81 per $100 of the payroll. In 1920 they were reduced to $5.34 for the stock companies and $4.28 for the State Insurance Fund and self-insurers. A further reduction was made in 1922 to $3.90 for the stock companies and $3.12 for the State and self-insurers. These rates prevailed until 1928 when, due to the increased number of accidents and their resultant medical and compensation costs, an increase in premium rates became necessary. The rates were increased from $3.90 to $5 for stock companies, and from $3.12 to $4.25 for the State and self-insurers. Reflecting this increase, the total premiums for the fiscal year 1927-1928 of $251,505 jumped to $335,830 for the fiscal year 1928-1929. Even this increase was not sufficient to cover the mounting costs of the greater number of accidents; so on January 1, 1930, the rates were again raised to $7 for stock companies and $5.95 for the State and self-insurers. Thus, in less than two years it has been found necessary to increase the compensation insurance rates approximately 80 per cent, and in this it may be well to remember that much, if not all, of the increase in the accident rate was due to the extension of mechanization in Utah's coal mines. Should accidents continue to increase in the same proportion as they have in the last four fiscal years, a further raise in the compensation premium rates is inevitable.

A review of the official reports of fatal accidents occurring in the coal mines of Utah shows conclusively that many of them could have been avoided. Likewise, the low accident records not only of certain of the coal mines of Utah, but of other large coal-mining companies throughout the United States with hazards fully as great as those common to Utah mines, give ample evidence that coal can be mined in Utah as well as elsewhere with low accident rates. Further, it is believed that should, say, 20 per cent of the present yearly cost of $375,000 for compensation insurance be expended for increased inspection and supervisory forces, closer supervision of working places, increased timber supplies, the installation and use of a safer type of equipment, the instruction and disciplining of employees, and a conscientious and continued effort put forth by the managers, supervisors, and working force, the accident rates of the coal mines of Utah can be reduced so that the compensation premium rates may be lowered to a point where the savings will be several times the money expended in those efforts.

Table 12 summarizes the fatal accidents by company and year.

DISCUSSION OF DATA PRESENTED

Falls

The Kinney Coal during its operating life, 1919-1925 had the unique record of not having a fatality from falls of coal or rock; however, this mine did have two fatalities, one from a mining machine in the underground working and one on the surface. The Chesterfield Coal Co. and its predecessor, the American Fuel Co. had no fatalities from falls of coal or rock during the period 1918-1929, and the Weber Coal Co. had but one fatality from falls of coal or rock during the period 1918-1929, inclusive.

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Table 12. - Fatal accidents in the coal mines of Utah by producing companies, 1918-1929

Name	1928	1929	1920	1921	1922	1923	1924	1925	1926	1927	1928	1929	Total
Blue Blaze Coal Co.[1]	-	-	-	-	-	-	-	-	1	-	1	-	2
Carbon Fuel Co.	-	2	1	-	2	1	5	1	1	-	-	-	13
Chesterfield Coal Co.[2]	1	-	2	-	1	-	-	-	-	-	1	1	6
Columbia Steel Corp.[3]	-	-	-	-	-	-	2	-	-	-	-	1	3
Grass Creek Fuel Co.	-	-	-	-	-	-	-	2	-	-	-	-	2
Independent Coal and Coke Co.	-	3	-	3	-	1	2	1	1	1	3	4	19
Kinney Coal Co.[4]	-	-	1	1	-	-	-	-	-	-	-	-	2
Liberty Fuel Co.	1	-	3	-	1	-	-	2	-	2	-	-	9
Lion Coal Co.	1	1	-	-	-	2	3	1	1	1	-	-	10
MacLean Coal Co.[5]	-	-	-	-	-	-	1	-	-	-	-	-	1
Milburn Mine Co.	1	-	-	-	-	-	-	-	-	-	-	-	1
Mutual Coal Co.[6]	-	-	-	1	1	2	-	2	1	3	1	-	11
National Coal Co.[7]	-	-	-	-	-	-	-	-	-	-	-	1	1
Pack and Allen Coal Co.[8]	-	-	-	-	-	-	-	1	-	-	-	-	1
Peerless Coal Co.	-	-	1	-	-	-	-	-	1	1	1	1	5
Royal Coal Co.[9]	-	-	1	-	-	1	2	-	-	1	-	-	5
Scofield Coal Co.	2	-	-	-	1	-	-	1	-	-	-	-	4
Spring Canyon Coal Co.	3	2	-	2	1	4	3	3	5	3	3	6	35
Standard Coal Co.	3	1	2	1	3	4	3	2	1	1	3	4	28
Sweet Coal Co.[10]	-	-	-	-	-	-	-	-	-	1	-	1	2
U. S. Fuel Co.	2	4	13	1	4	3	3	4	4	2	3	7	50
Utah Fuel Co.	5	14	10	6	8	5	176	4	5	10	2	7	252
Weber Coal Co.	-	-	1	-	-	-	-	-	-	-	-	1	2
Total	19	27	35	15	22	23	200	24	21	26	18	34	464

1 - Consumers Mutual Coal Co. prior to 1927. Commercial production since 1926. 2 - American Fuel Co. prior to 1923. 3 - Commercial production since 1923. 4 - Ceased production in 1925. 5 - Commercial production since 1923. 6 - Commercial production since 1921. 7 - Commercial production since 1928. 8 - Wagon mine. 9 - Cameron Coal Co. prior to 1924. 10 - Commercial production since 1927.

Haulage

Companies which had no fatalities from mine cars or locomotives underground are the Grass Creek Fuel Co. during the period 1918-1929, inclusive, the Kinney Coal Co. through its entire existence, 1919-1925, inclusive, the MacLean Coal Co. from 1923-1929, inclusive, the National Coal Co. in the two years 1928 and 1929, the Peerless Coal Co. from 1918 to 1929, inclusive, the Sweet Coal Co. 1927-1929, inclusive, and the Weber Coal Co. 1918 to 1929, inclusive.

Explosions

During the period 1918 to 1929, inclusive, mines had explosions of gas or dust as follows: Carbon Fuel Co. in 1929 with one death and in 1924 with five fatalities; Spring Canyon Coal Co. in 1924 with two fatalities and in 1929 with one death; the Standard Coal Co. in 1923 with two fatalities; and the Utah Fuel Co. in 1924 with 171 fatalities.

After the Winter Quarters disaster of May 1, 1900, with its 200 fatalities, very rigid precautionary measures were adopted and kept in effect, in consequence of which nearly 24 years elapsed before the occurrence of another major disaster; however, as early as 1915 or 1916 there were numerous relaxations in the precautionary methods and practices, and the failure to have had disasters long before those of 1924 at Castlegate and at Rains can be considered as merely fortunate. In a similar manner, several years have now elapsed since the Castlegate and Rains disasters, and there is no question that many unsafe practices and methods are in effect in Utah's coal mines; in particular, much unsafe electrical equipment is being placed at gassy and dusty faces, the inevitable result of which will unquestionably be the occurrence of explosions and fires unless drastic steps are taken towards enforcing adequate safeguarding measures.

Explosives

The companies that had fatalities from explosives were The Independent Coal & Coke Co. in 1927, one death; the Liberty Fuel Co. in 1918, one fatality; the Spring Canyon Coal Co. in 1923, one death; the Standard Coal Co. in 1919, one fatality, and the Utah Fuel Co. in 1922, two fatalities. The record of the United States Fuel Co. with five large mines producing during much of the period 1918 to 1929, and with no fatalities from explosives, is a definite indication of the safety of the blasting system used exclusively by this company; all shots are fired electrically from the surface when all persons, including the shotfirer, are out of the mine. This system is also used by the mine of the Columbia Steel Corporation which has had no fatalities from blasting during its existence from 1923 to 1929, and by the Peerless Coal Co. and the Royal Coal Co., both of which operated from 1918 to 1929 without a fatality from blasting.

Electricity

Fatalities from electricity occurred as follows: Carbon Fuel Co. in

1920, one death on the surface; Chesterfield Coal Co., one death in 1922 on the surface; the Independent Coal & Coke Co. in 1929, one death on the surface; the Liberty Fuel Co. in 1920, one death on the surface; the Mutual Coal Co., one death underground in each of the years 1922, 1925, and 1926; the Royal Fuel Co., one death underground in 1927; the Spring Canyon Coal Co., one death on the surface in 1921 and one underground in 1929; the United States Fuel Co., one death on the surface in 1918; the Utah Fuel Co., one death underground in 1921, and the Weber Coal Co., one on the surface in 1920. Of the 13 deaths from electricity at Utah's coal mines during the period 1918 to 1929, seven were on the surface, indicating laxity in surface electrical installations. Ten of the 23 coal-mining companies of Utah had fatalities from electricity during the period 1918 to 1929.

Mining Machines

Mining machines caused the only underground fatality in the career of the Kinney Coal Co.; this death occurred in 1921, and the mine was closed in 1925. Other fatal accidents from mining machines were one in 1918 in a mine of the Spring Canyon Coal Co.; one in 1928 in a mine of the Standard Coal Co.; one in 1919 and one in 1920 in mines of the United States Fuel Co.; and two in 1919 and one in 1921 in mines of the Utah Fuel Co.--a total of eight for the period under review. Most of these accidents could have been avoided by adequate guarding and more careful use of mining machines; the latter for instance, the practice of leaving but one man on a coal-cutting mining machine, in vogue in some of Utah's coal mines, is so manifestly dangerous that it should not be allowed. Accidents of all classes caused by various types of mining machines can be expected to occur and probably with increasing frequency and severity unless the most rigid of precautionary measures are adopted and maintained in the mechanization of underground loading and other processes now being extensively introduced into the coal mines of Utah, as well as of other States.

Surface Fatalities

On the surface, mine cars and locomotives caused deaths as follows: The Chesterfield Coal Co., one in 1918; Lion Coal Co., two in 1924; Peerless Coal Co., one in 1928; Spring Canyon Coal Co., one in 1919; and the Standard Coal Co., one each in 1920, 1922, 1928, and 1929; a total of nine. Inasmuch as most of these fatalities were in connection with walking on surface inclines, it would appear advisable, if practicable, to make adequate provision for safe travel ways for men rather than to allow them to travel on or alongside of these haulage tracks on inclines. These surface inclinos also have a definite hazard in the man-trips which consist merely of empty cars, and essentially without taking any precautions other than would be taken with the same number of cars loaded with coal; it would appear advisable to have specially constructed man cars or man trips with all available safeguards and precautions; if this is not done, major disasters are likely to occur in connection with man trips on those inclines. As previously indicated, seven persons were killed on the surface by electricity, as well

as six others in underground workings, indicating laxity in both the installation and use of electricity in and around Utah coal mines.

Mining companies with no surface fatalities during the period of this study are the Blue Blaze Coal Co., Columbia Steel Corporation, Grass Creek Coal Co., MacLean Coal Co., Mutual Coal Co., National Coal Co., Royal Coal Co., and the Sweet Coal Co. Surface accidents at coal mines in Utah, as well as elsewhere, are caused largely by engineering failures or inefficiencies; this is also the case with accidents from mine cars or locomotives, from electricity, from railroad cars and locomotives, and from a number of other sources of accident occurrence on the surface at coal mines. The engineer who neglects safety in the design and operation of surface structures and equipment has done a poor job, unfortunately, safety is often neglected in tipples, mine railroad yards, hoisting or other haulage arrangements, and at other places on the surface.

Table 13 lists the fatality rates per million tons of coal produced for certain Utah companies compared with the rate for all bituminous mines in the United States.

Table 13. - Relative fatality rates per million tons produced in the coal-mining companies of Utah

Period	Name	Company rate	Rate for all coal mines of Utah	Rate for all bituminous mines of the United States
1923 - 1929	Columbia Steel Corporation	1.3	10.4	3.5
1918 - 1929	Royal Coal Co.	2.5	8.0	3.5
1918 - 1929	Peerless Coal Co.	2.9	8.0	3.5
1919 - 1925	Kinney Coal Co.	3.2	10.2	3.5
1926 - 1929	Blue Blaze Coal Co. . . .	3.4	5.1	3.4
1918 - 1929	Independent Coal & Coke Co.	3.6	8.0	3.5
1918 - 1929	United States Fuel Co. . .	4.2	8.0	3.5
1918 - 1929	Liberty Fuel Co.	4.3	8.0	3.5
1918 - 1929	Scofield Coal Co.	4.7	8.0	3.5
1918 - 1929	Chesterfield Coal Co. . .	4.8	8.0	3.5
1918 - 1929	Weber Coal Co.	5.0	8.0	3.5
1918 - 1929	Carbon Fuel Co.	5.9	8.0	3.5
1918 - 1929	Lion Coal Co.	6.2	8.0	3.5
1923 - 1929	MacLean Coal Co.	6.4	10.4	3.5
1927 - 1929	Sweet Coal Co.	6.5	5.2	3.3
1918 - 1929	Standard Coal Co.	6.9	8.0	3.5
1918 - 1929	Spring Canyon Coal Co. . .	7.1	8.0	3.5
1918 - 1929	Grass Creek Fuel Co. . . .	7.4	8.0	3.5
1928 - 1929	National Coal Co.	8.2	5.2	3.3
1921 - 1929	Mutual Coal Co.	8.3	9.0	3.5
1918 - 1929	Utah Fuel Co.	17.8	8.0	3.5

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[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

The fatality rates per 1,000,000 tons produced for the coal-mining companies of Utah compared with like rates for all of the bituminous mines of the United States show four companies with better rates than those of the bituminous coal industry as a whole, and one equally as good. The majority of Utah companies present accident rates considerably above those prevailing for the whole country, although the thick beds of coal in Utah and the high tonnage per man employed gives its coal mines a distinct advantage in accident rates on a tonnage basis.

Compared with the rates for all coal mines of the State, only three companies fail to have a better record than the State rates. The State rate, influenced by the Castlegate and Rains disasters for the period including 1924 is abnormally high, as will be seen by comparing rates for periods excluding 1924 where differences of three to five points are shown.

Table 14 lists the coal produced per fatality in certain Utah and in all bituminous mines.

Table 14. - Relative tonnages produced per fatality for the coal mines of Utah

Period	Name	Company tonnage	All coal mines of Utah, tonnage	All bituminous mines of the United States, tonnage
1923 - 1929	Columbia Steel Corporation .	763,380	95,472	285,578
1918 - 1929	Royal Coal Co.	386,850	124,736	285,001
1918 - 1929	Peerless Coal Co.	342,597	124,736	285,001
1919 - 1925	Kinney Coal Co.	306,656	97,127	279,251
1926 - 1929	Blue Blaze Coal Co.	298,905	193,279	293,523
1918 - 1929	Independent Coal & Coke Co.	270,920	124,736	285,001
1918 - 1929	United States Fuel Co. . . .	235,032	124,736	285,001
1918 - 1929	Liberty Fuel Co.	229,720	124,736	285,001
1918 - 1929	Scofield Coal Co.	212,045	124,736	285,001
1918 - 1929	Chesterfield Coal Co. . . .	208,195	124,736	285,001
1918 - 1929	Weber Coal Co.	198,820	124,736	285,001
1918 - 1929	Carbon Fuel Co.	168,867	124,736	285,001
1918 - 1929	Lion Coal Co.	160,699	124,736	285,001
1923 - 1929	MacLean Coal Co.	156,115	95,472	285,578
1927 - 1929	Sweet Coal Co.	152,787	189,242	299,856
1918 - 1929	Standard Coal Co.	144,060	124,736	285,001
1918 - 1929	Spring Canyon Coal Co. . . .	139,830	124,736	285,001
1918 - 1929	Grass Creek Fuel Co.	133,516	124,736	285,001
1928 - 1929	National Coal Co.	121,200	121,902	301,194
1921 - 1929	Mutual Coal Co.	119,101	109,932	282,077
1918 - 1929	Utah Fuel Co.	55,914	124,736	285,001

The tonnage produced per fatality for five of the coal companies of Utah are above the average for all the bituminous mines of the United States, whereas the balance of the companies show tonnages which are far from favorable. Compared with tonnages for all the coal mines of the State, however, only three companies fail to equal the State average. Here again the influence of the disaster of 1924 is apparent in the State tonnages where that year is included in the periods considered.

Table 15 gives the coal mined per employee in certain Utah mines and in all bituminous mines.

Table 15. - Yearly tonnage produced per employee for the coal mines of Utah

Period	Name	Company tonnage	All coal mines of Utah, tonnage	All bituminous mines of United States, tonnage
1923 - 1929	Columbis Steel Corporation .	1,734	1,197	888
1918 - 1929	Independent Coal & Coke Co.	1,393	1,174	832
1918 - 1929	Liberty Fuel Co.	1,389	1,174	832
1918 - 1929	United States Fuel Co. . . .	1,365	1,174	832
1928 - 1929	National Coal Co.	1,247	1,320	978
1927 - 1929	Sweet Coal Co.	1,217	1,354	939
1918 - 1929	Peerless Coal Co.	1,204	1,174	832
1926 - 1929	Blue Blaze Coal Co.	1,139	1,324	946
1918 - 1929	Royal Coal Co.	1,111	1,174	832
1921 - 1929	Mutual Coal Co.	1,100	1,146	822
1918 - 1929	Spring Canyon Coal Co. . . .	1,096	1,174	832
1918 - 1929	Standard Coal Co.	1,086	1,174	832
1918 - 1929	Lion Coal Co.	1,061	1,174	832
1918 - 1929	Carbon Fuel Co.	974	1,174	832
1918 - 1929	Chesterfield Coal Co. . . .	965	1,174	832
1919 - 1925	Kinney Coal Co.	953	1,096	741
1918 - 1929	Utah Fuel Co.	947	1,174	832
1923 - 1929	MacLean Coal Co.	918	1,197	888
1918 - 1929	Scofield Coal Co.	888	1,174	832
1918 - 1929	Weber Coal Co.	746	1,174	832
1918 - 1929	Grass Creek Fuel Co.	745	1,174	832

The tonnages produced per employee by the coal-mining companies of Utah have always been high, and in only three instances of the periods studied has production fallen below the averages for all the bituminous mines of the country. Less than one-half of the companies, though, showed tonnages equal to the averages for Utah.

Table 16 gives the fatality rates per 1,000 employees for certain Utah companies and for all bituminous mines.

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Table 16. - Relative fatality rates for thousand employees for the coal mining companies of Utah

Period	Name	Company rate	Rate for all coal mines of Utah	Rate for all bituminous mines of the United States
1923 - 1929	Columbia Steel Corp. . . .	2.2	12.5	3.1
1918 - 1929	Royal Coal Co.	2.8	9.4	2.9
1919 - 1925	Kinney Coal Co.	3.1	16.7	2.6
1918 - 1929	Peerless Coal Co.	3.5	9.4	2.9
1918 - 1929	Weber Coal Co.	3.7	9.4	2.9
1926 - 1929	Blue Blaze Coal Co. . . .	3.9	6.8	3.2
1918 - 1929	Scofield Coal Co.	4.1	9.4	2.9
1918 - 1929	Chesterfield Coal Co. . .	4.6	9.4	2.9
1918 - 1929	Independent Coal & Coke Co.	5.1	9.4	2.9
1918 - 1929	Grass Creek Fuel Co. . . .	5.6	9.4	2.9
1918 - 1929	Carbon Fuel Co.	5.7	9.4	2.9
1923 - 1929	MacLean Coal Co.	5.8	12.5	3.1
1918 - 1929	United States Fuel Co. . .	5.8	9.4	2.9
1918 - 1929	Liberty Fuel Co.	6.0	9.4	2.9
1918 - 1929	Lion Coal Co.	6.6	9.4	2.9
1918 - 1929	Standard Coal Co.	7.5	9.4	2.9
1918 - 1929	Spring Canyon Coal Co. . .	7.8	9.4	2.9
1927 - 1929	Sweet Coal Co.	7.9	7.1	3.1
1921 - 1929	Mutual Coal Co.	9.2	10.4	2.9
1928 - 1929	National Coal Co.	10.3	6.8	3.2
1918 - 1929	Utah Fuel Co.	16.9	9.4	2.9

The fatality rates per 1,000 employees for the coal-mining companies of Utah compared with the same rates for all bituminous mines of the United States for like periods reveal only two Utah coal-mining companies--the Columbia Steel Co. and the Royal Coal Co.--having rates equal to or better than the rate for the United States. These two instances prove, however, that it is possible to mine coal in Utah with as low a fatality rate as elsewhere in the United States, and these mines afford records which the other coal-mining companies can, if they will, duplicate or better.

The records of nearly one-half of the companies show rates more than double those for the bituminous mines of the United States, and with several of the companies the rates are three or four times as high.

The company rates compared with those for all the coal mines of Utah are in most cases low, but this is due wholly to the fact that the years considered in this study include 1924 when the Castlegate and Rains disasters with 177 fatalities occurred, and this gives a State rate decidedly higher than normal.

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UTAH COAL-MINING REGULATIONS

Utah's coal-mining regulations are generally held to be the most drastic in the United States, if not in the world, and embody many of the most up-to-date policies for safeguardihg mines and employees. These regulations have either been framed by the coal operators or have been adopted only after the written approval of the operators has been received. Not only have the Utah coal operators had a deciding voice in the framing of every State regulation under which they are working, but they may at any time have any State regulations as to individual or unusual conditions suspended or entirely revoked or changed, by presenting to the State Industrial Commission a reasonable showing why such suspension or change should be made.

Utah's original mining law was altered little, if at all, between 1900 and 1917, when the act was passed creating the State Industrial Commission and placing the inspection of coal mines thereunder. In 1918, an agreement was made with the Federal Government, whereby the latter furnished a mining engineer who acted as chief mine inspector representing both the State and the Federal Government. Shortly after this act became effective, a committee of five coal operators and mining engineers, representing the larger as well as smaller producers, together with the chief mine inspector, the State coal-mining inspector, and three electrical engineers, was appointed to frame detailed regulations for Utah's coal mines. Very detailed, and in some respects radical, regulations were adopted and placed in pamphlet form; the regulations became effective on September 1, 1920.

Under these regulations, detailed safety rules were issued regarding most of the operations in a coal mine; some of the outstanding features were the following: Every mine was required to have a checking system which would account for the whereabouts of every man in the mine and especially show when all were out of the mine. All mines, whether gassy or nongassy, were required to have a regular fire boss's inspection before the shift entered the mine. All mines were required to have first-aid supplies and equipment and first-aid stations underground; where more than 50 men were employed underground, the mines were required to have a corps of men trained in first aid and in mine rescue, and to have rescue apparatus available. Travel ways separate from haulage roads were required where the dip of the coal exceeded 10°. Hoisting enginemen were required to have a health certificate. Mining machines must have guards over the bits; numerous provisions were also made as to guarding gears, and other features of the tipple as well as of underground machinery. Sprinkling was required except in places naturally wet or made wet by the use of steam; water hose would be furnished any face worker who requested it. In addition, there were fairly complete provisions as to ventilation, use of electricity, safety lamps, timbering, the use, handling, and storage of explosives, and as to other operations.

After the Federal leasing act became effective in 1921, many of the Utah coal companies took Federal leases, operating the leased land through existing mines, thereby making the Federal leasing regulations applicable to such mines. These Federal regulations are held to be somewhat radical, yet they, as well as Utah's regulations, are now applicable to Utah coal mines producing much of the coal of the State.

The Castlegate explosion in March, 1924, created a feeling that some of the optional or provisional requirements of both the Utah regulations and of the Federal leasing regulations should be made obligatory. Therefore, at the request of the State Industrial Commission of Utah, the Utah Coal Mine Inspection force and some engineers of the United States Bureau of Mines suggested some modifications of the Utah coal-mining regulations which were submitted to a committee of Utah coal operators. This committee, composed of members appointed by each of the 18 main operating coal companies of Utah and representing more than 95 per cent of the total State production, consisted of general managers, general superintendents, chief engineers, and mine superintendents, every one with many years of mining experience and undoubtedly the men best qualified to safeguard the interests of their employers. This committee accepted without change six of the proposed new regulations and accepted the other five after they were somewhat amended. In turn, the State Industrial Commission accepted the findings of the coal operators' committee, including the changes made by the committee. The revised so-called drastic regulations adopted in Utah, in 1924, in this way received the written approval of every large coal-mining company in Utah before they were adopted. Under these new regulations, Utah coal mines were the first in the United States to require rock-dusting and to use rock-dust barriers; to exclude black powder and use nothing but permissible explosives in the shooting of coal; to exclude fuse and squibs and have all blasting done by electricity, (even then no blasting may be done with the shift in the mine); to exclude open lights and allow nothing but permissible closed electric lamps for the miners and permissible magnetically locked flame safety lamps for inspection work, the requirement extending to so-called nongassy as well as to gassy mines; to have water piped to every working face with use of water hose at and around face by miners; to compel all coal cutting to be done with simultaneous use of water to kill the bug dust; and to prohibit removal of methane accumulations while the working shift is in the mine.

OBSERVANCE OF MINING REGULATIONS

Although several of the coal companies have tried to adhere to the spirit as well as the letter of the law and the regulations under the law, and in many instances have excellent and somewhat expensive safety methods, practices, and equipment not required by the law, there are undoubtedly mines where not even the letter of the law has been obeyed. Individual miners, and in some instances mine officials tell of mines where smoking has been done underground with more or less the consent of the operator. Open torches for welding, bonding, and other work have been used more or less generally and with the shift in the mine. Dynamite has been used ostensibly for blasting rock, but actually in shooting coal in some instances. Gas accumulations have been moved with the shift in the mine, and in some cases, sprinkling has been used to waft the gas into the ventilating current, notwithstanding the fact that this very poor practice resulted in the Castlegate explosion of 1924 with its 171 fatalities. More or less open abandoned places are unprotected by rock-dust, rock-dust barriers, sprinkling, or sealing. Rock-dusting, if done at all, is in many mines done in a perfunctory manner with insufficient dust or with poor distribution, or is not renewed when

covered with coal-dust in explosive proportions. Use of water on cutting machines, or on coal at faces or sprinkling of nonrock-dusted haulage ways, returns, room necks, and other places is done inefficiently, and dry coal-dust is freely found in the lower part of rooms, in air courses, crosscuts, open abandoned workings, and on many nonrock-dusted haulage roads. Explosives are taken into the mine in excessive quantities and are not properly protected while in the mine, and are used in such large quantity per hole that the permissible or safety feature is destroyed. Flame safety lamps, supposed to be kept locked when underground, are opened underground to be relighted because the relighting devices are not kept in working order. While verification of some of these statements would be difficult, there is little doubt that there is foundation for them and good reason to believe that other violations of the regulations are more or less current practice.

On the other hand, many mining men who were skeptical of rock-dusting, yet complied with the regulations, now are dusting far beyond the requirements of the State and the cost is found to be less than that of adequate sprinkling. Some mines are carrying the rock-dusting so far into the interior of the mine that the barrier requirement is being waived. Of those who objected to the use of water on the mining-machine cutter chain, few now would abandon this practice even if permitted to do so. The order eliminating black powder affected very few mines. While black powder still has some advocates, its loss has caused so little inconvenience as to be negligible, and much of this inconvenience is a state of mind rather than a matter of fact. The elimination of fuse and squibs and of blasting during the shift was an actual benefit to every person concerned; moreover, it applied to but a small portion of Utah's coal mines, inasmuch as more than half of Utah's tonnage had been on the new basis for several years. The requirement of closed electric safety lamps for all mines, including the so-called nongassy mines, has been a decided practical success. Disapproval of the new system is negligible, as is shown from the fact that when a hearing was called by the State Industrial Commission in February, 1925, to discuss the advisability of modifying the closed-light order, no advocate of the open light appeared, and the hearing developed into a "love feast" in favor of closed light for all mines.

Utah, for more than a quarter of a century, has used widespread sprinkling of workings, probably to a greater extent than any other coal-mining region in the world, and the new regulations as to sprinkling merely made obligatory practices already largely in effect but chiefly optional. However, the new regulations caused much more extensive, and at the same time much more intensive use of water, especially at and near the faces; so assiduously was the water applied that the wet slack became difficult to screen, and as a consequence some companies now are neglecting the sprinkling in order better to satisfy consumers as to coal screening. Other mines, however, complying with the spirit as well as the letter of the law, use the water, regardless of its effect on coal screening and sizing. Two of the 11 so-called drastic regulations have been rather freely disobeyed or disregarded; one of these is the requirement that methane accumulations shall not be moved while the shift is in the mine; the other is that before track is pulled out of rooms, air courses, or other workings, loose coal and coal-dust must be loaded out. The moving of methane accumulations during the shift has been almost a general practice in

coal mining; hence, it is difficult to change. However, it is very dangerous, has resulted in innumerable mine disasters, and, notwithstanding the opinion to the contrary, it is absolutely unnecessary. The practice certainly should be stopped.

The drastic regulations as to Utah coal mines have been in effect several years; while they caused additional expense in their initiation, there is every reason to believe that if enforced impartially the net result over a period of years would be a monetary saving rather than an extra charge. However, with but one State inspector for mines with about 3,500 men producing approximately 5,000,000 tons annually, under conditions so inherently dangerous as those in Utah's coal mines, it can readily be seen that the State inspection force is entirely inadequate; and in time the regulations will be a joke except with those operators who really are progressive and possessed of a conscience. The regulations are up-to-date, safe, sane, practical, even economical, but they should be strictly enforced on all engaged in coal mining in Utah, and thereby made a protection to those operators who really desire to have their mines safe, as against those who would, if allowed, sacrifice safety of mine and men to expediency or to their own opinions.

SUGGESTED IMPROVEMENTS IN SAFETY MEASURES

Many of the smaller mining companies in Utah have their individual safety specialties; on the other hand, some mines do as little as possible and yet show enough compliance with the regulations to avoid the disapproval of the inspector. The numerous factors that tend toward making it difficult to operate with safety in the coal fields of Utah, together with the fact that the State's coal-mine accident record, both as to individual accidents and as to disasters, is decidedly bad notwithstanding the many excellent safety practices, laws, and regulations in effect, indicate that much yet remains to be done to safeguard Utah's mines and miners; therefore, a few suggestions as to possible improvements are here given.

Mining Methods

1. There appears to have been little or no alteration or improvement in the general system of mining in Utah during the past 20 years; as a result, there is too much interconnection of mines, too much interconnection of different mine levels, with far too many open nonworking rooms, entries, and other places. The remedies are: Adoption of panel systems with but two, three, or four openings from each panel, separating each panel from all others in so far as possible. Room pillars should be pulled as soon as rooms reach their distance; where this can not be done, the open nonworking rooms, and as much as possible contiguous entry, pillar, and other open nonworking territory, should be sealed. In pulling pillars, greater care should be taken to establish and maintain a predetermined pillar line, and pillars should be pulled clean even when to do so may be costly. Entries from which rooms are turned should be little, if any, over 1,000 feet long; otherwise maintenance is likely to become costly, and the haulage ways, manways, and other openings would be dangerous before the region is worked out. Where nonworking territory must be kept

open, whether rooms, entries, or pillars, this open nonworking territory should be kept well ventilated, well watered, preferably by water sprays running continuously, or rock-dusted; in any event it should be well protected by rock-dust barriers as open nonworking territory is very little, if any, less dangerous than are working faces. No nonworking territory should be left open if it can be sealed.

Haulage

2. Haulage casualties occur much too frequently in Utah coal mines; a few possible precautionary measures are:

Long, steep inclines on which men are hauled should be equipped with safety man trips, so that if a rope, coupling, or drawbar should break, automatic devices will stop the trip. If this is not done, it is only a question of time until there will be very serious disasters on these inclines.

Bare trolley wires carrying 250 to 500 volts direct current should be kept well boxed where the wire is less than 7 feet above the rail; this is usually held to be impracticable, but some metal mines have miles of effectively boxed trolley lines.

Haulage-entry grading is too frequently held to be dead work and a luxury; it is necessary, pays well, and is a wonderful safety precautionary measure.

In general, the pit cars and grades are far too heavy and dangerous for animal haulage; mine workings should be planned to use permissible storage-battery locomotives for gathering purposes. It, however, seems to be anything but safe practice to put trolley locomotives, even the crab or reel type, to the face of workings known to give off methane; this type of locomotive should not go into the rooms or pillars of workings with dust as inflammable as that found in Utah's mines, even if no methane is known in such places.

The open-end gate type of pit car is completely out of date in efficient coal mining, is wasteful, and is decidedly dangerous; nothing but the no-gate type of pit car should be allowed in Utah coal mines.

Blasting

3. Blasting practice in Utah is decidedly good, but is deficient in some respects.

No explosive should be allowed inside when the shift is in the mine. The shot-firers, who would thus carry in the explosive and load and tamp as well as fire all shots after departure of the working shift, should be required to have certificates of competency after having passed an examination, which should be at least as searching as the examination for fire bosses.

While nothing but permissible explosives are supposed to be used for shooting coal in Utah's coal mines, some dynamite, ostensibly for shooting

rock, is being used to shoot coal or coal and rock together. Dynamite should be wholly eliminated from Utah's coal mines because permissible explosives can be obtained which, when used correctly, will bring down any rock or any coal encountered.

Permissible explosives are not held to be safe or permissible when more than $1\frac{1}{2}$ pounds, or approximately 3 sticks, are charged into a hole; as a matter of fact, as many as 6 sticks, and more, are being used in Utah mines because of lack of efficient supervision, or in some instances because of failure to drill enough holes. This use of excessive permissible explosive per hole is dangerous and is also certain to give an unnecessary amount of fines and shattered coal.

There is a possibility that where the surface electric shooting system is in use, the shot-firers do not go to the surface to fire the shots, as required. There should be established some method which will insure that the shot-firers do go to the surface before or while firing the shots; otherwise the system will be ineffective.

Where the surface shooting system is in use, the main shooting switch at the surface should be so constructed that electrical contact is automatically maintained for but a fraction of a second; there should also be definite assurance that the main shooting switch will not be thrown in a second time unless there has been complete underground inspection and rearrangement of underground shooting switches and other accessories.

Ventilation

4. Ventilation practice in Utah coal mines is above the average, but there are serious defects:

Doors on haulage roads indicate a poor ventilation system; this is likely to be decidedly serious in gassy mines. Ventilation should be planned so that overcasts, regulators, and other means will eliminate doors from haulage roads, or even from most of the main manways.

Too many doors, stoppings, and other appliances in Utah mines of a more or less permanent nature are constructed of brattice cloth, wood, or other inflammable material.

Small electrically driven booster fans with tubing are being used increasingly in coal mines to ventilate dead ends. This indicates a failure or breaking down of the general ventilation system. Electric motors are very dangerous in gassy mines, and unless great care is exercised in placing them, and also in starting and stopping them, a disaster is likely to occur.

Air courses that are not haulage ways (and some air courses that are haulage ways) are allowed to become clogged, or nearly so, with rock or coal

falls, water, or debris to such an extent that air flow is greatly impeded, if not practically stopped. One Utah company, by cleaning a few of its airways, increased the mine air flow by more than one-third.

Search for matches and patent lighters is not made as frequently or as rigidly as this safety feature deserves; it is said that in some coal mines, the mine officials wink at smoking underground. This is decidedly dangerous, besides being a defiance of the law. Several recent coal-mine explosions in closed-light mines have been caused by miners smoking; matches are often found on the bodies recovered.

It seems probable that flame safety lamps are being opened underground and relighted by open-flame matches; in several instances, flame safety lamps have been found with a defective relighting device; hence, they could have been relighted underground only by opening the lamp and lighting with a match. Some system should be devised at each mine to have all the flame safety lamps inspected at least once each week by some reliable person other than the user, even though the useris the fire boss, foreman, superintendent, or other official.

Electricity

5. Undoubtedly, electricity constitutes the greatest probable source of danger in coal mines generally, underground electrical installation are anything but safely or efficiently placed or maintained.

Few underground workers, whether miners or officials, are sufficiently well informed regarding electricity to be able to give an adequate opinion as to the safety of electric wiring, hoisting, pumping, haulage, coal-cutting, or other installations; this is particularly true of inspectors, whether working for the Federal Government, State, or the companies themselves. A competent electrical inspector should be provided by the State, and, if possible, he should have experience in coal mines as well as a thorough familiarity with electricity and its dangers.

In most coal mines, electric wiring is neglected in back entries or out-of-the-way places, frequently also, near the faces and in main haulage or travel roads. This is poor policy, as defective wiring in mines is even more dangerous than in city houses; as a matter of fact, mine wiring should be more carefully placed and maintained than surface or house wiring. Utah's coal mines have much defective wiring; this is true especially of shooting wires in rooms and in entries near the face.

It is inconsistent to require closed lights for miners yet allow use of the open, nonpermissible, electric coal-cutting and drilling equipment at faces. It would be advisable to confine future installations of electrical equipment, even in nongassy mines, to the permissible variety. Where methane is known to occur, if only in small quantity, nonpermissible electrical machinery should be withdrawn and permissible equipment substituted and maintained in permissible condition.

The open bare places on power wires to fasten the nips of the mining-machine trailing cable should be kept covered and held outside the last cross-cut. Even then they should be placed as near the floor as possible, rather than near the roof. A much better practice, however, is to have enclosed connectors and thus eliminate all danger from contact with bare spots or timbering or from open arcing or mechanized mining.

Supervision

6. Supervisory force in coal mines of Utah, as well as in most other coal-mining States, appears not to be as efficient in safety as it should, so the following suggestions are offered:

(a) Utah should have at least two general coal-mine inspectors and one inspector of coal-mine electrical equipment; these inspectors should be on a civil-service basis, should be appointed after having passed a difficult examination, both physically and mentally; and should be reexamined about every five years.

(b) All shot-firers, fire bosses, mine foremen, mine superintendents, and general superintendents should be required to have certificates of competency after having passed an examination involving both mental and physical tests as to knowledge of mines and mining and ability to do the necessary underground work. These certificates should be renewable every five years after a similar examination has been passed. This seems drastic, but it might aid in eliminating from official positions many men unfit, either physically or mentally, to hold those positions.

CONCLUSION

In general, safety practices in Utah's coal mines are better than those observed in coal mines of other States, but Utah's coal mines have so many inherent natural hazards that they demand much better safety practices than mines of other States. There should certainly be no letting down of the bars; rather, those bars should be built higher and made more difficult to get through or around or over.

I. C. 6533 November, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

FELDSPAR GEMS

(AMAZON STONE, MOONSTONE, SUNSTONE, ETC.)

BY

I. Aitkens

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

FELDSPAR GEMS 1/

(Amazon stone, Moonstone, Sunstone, et cetera)

By I. Aitkens 2/

CONTENTS

INTRODUCTION

The group of gems included under the general name "feldspar" occupies only a minor place in jewelry. Feldspars, with the possible exception of quartz, are the most important of rock-forming minerals, but most occurrences are of quite ordinary varieties, without striking color and unattractive. Few specimens are suitable for use as gem stones, but occasional crystals that exhibit unusual brilliance or play of color are cut into various shapes for personal adornment or for architectural decoration. The best known of the feldspar gems, and probably the most important in point of production, is the moonstone. Amazon stones, labradorite, sunstone, and a number of less common varieties have been produced in various parts of the world.

The present paper discusses only the gem varieties of feldspar. Data

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1/ The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6533."

2/ Rare metals and nonmetals division, U. S. Bureau of Mines.

as to commercial grades are summarized in another publication[3/] of the Bureau of Mines.

DESCRIPTION AND PROPERTIES

The feldspar group is a family of closely related minerals, all of which are composed of compounds of silica and alumina with varying amounts of soda, potash, or lime. The more important members of the feldspar group are orthoclase, microcline, albite, oligoclase, labradorite, and anorthite. The formulas for the principal varieties of feldspar are as follows:

Orthoclase and microcline	Potassium-aluminum silicate	$KAlSi_3O_8$
Albite	Sodium-aluminum silicate	$NaAlSi_3O_8$
Oligoclase	Aluminum-calcium and sodium silicate	$3NaAlSi_3O_8 \cdot 1CaAl_2Si_2O_8$
Labradorite	do	$1NaAlSi_3O_8 \cdot 3CaAl_2Si_2O_8$
Anorthite	Aluminum and calcium silicate	$CaAl_2Si_2O_8$

In crystallization, orthoclase is monoclinic, while the other feldspars are triclinic. Twinning is very common, and all varieties show cleavages in two directions. The hardness is about 6 and the specific gravity ranges from 2.5 to 2.8.

Feldspars are colorless, white, pale yellow, green, or reddish. Most of them, however, are without striking color or optical effects. They have a vitreous to pearly luster and are transparent to opaque. Although the mean index of refraction varies from 1.52 to 1.58, the double refraction is low, ranging from 0.005 to 0.012. Dispersion in the feldspars is also low, albite being 0.012. All the feldspars are biaxial; albite and labradorite are optically positive but the other varieties are negative.

Moonstone

The gem varieties of orthoclase are transparent and colorless and are called adularia, or moonstone when opalescent. However, there are moonstone varieties of albite and oligoclase also.

Amazon stone

The bright green variety of microcline is called amazon stone or amazonite. This varies from a light verdigris-green to a deep blue green and finds wide application on account of its pleasing color, which is very similar to jade.

3/ Bowles, Oliver, and Lee, C. V., Feldspar: Inf. Circ. 6381, Bureau of Mines, October, 1930, 21 pp.

Sunstone or Aventurine

Sunstone, or aventurine oligoclase is reddish, with bright yellow or red reflections. These reflections are caused by iron oxide and foreign inclusions which give the mineral a golden shimmer and sparkle.

List of Varieties

Besides the more important members already given, the feldspar group of gems is divided into many classes under the different classifications and divers names have been given them. The following alphabetical list gives the gem and mineral names of the feldspar group:

Gem name	Common name or description
Adularia	Orthoclase (feldspar).
Albite	Silicate of aluminum and soda (feldspar).
Albite moonstone	Iridescent albite.
Amazone stone	Green microcline (feldspar), silicate of aluminum and potassium.
Andesine	Silicate of aluminum, sodium, and calcium (feldspar).
Adventurine feldspar	Sunstone.
Bemiscite	Salmon-colored feldspar from Bemis, Maine.
Bull's eye	Labradorite with a dusky sheen.
Cassinite	Pearly, bluish green aventurine feldspar from Delaware County, Pa.
Ceylon opal	Moonstone.
Changeant	Labradorite.
Chesterlite	Microcline feldspar from Chester County, Pa.
Delawarite	Aventurine feldspar from Delaware County, Pa.
Feldspar sunstone	Sunstone.
Fish-eye	Moonstone.
Graphic-granite	Pegmatite composed of quartz and feldspar so arranged as to simulate writing.
Heliolite	Sunstone (feldspar).
Labrador feldspar	Labradorite.
Labrador spar	Labradorite.
Labrador stone	Labradorite.
Labradorite	Feldspar, silicate of aluminum, sodium and calcium.
Leelite	Deep flesh-red orthoclase (feldspar) having a waxy luster.
Lennilite	Greenish feldspar from Lenni Mills, Delaware County, Pa.
Microcline	Potash feldspar in triclinic crystals, silicate of aluminum and potassium.
Moonstone	Feldspar (usually oligoclase or the adularia variety of orthoclase) showing a pearly opalescence; also commonly but erroneously applied to some white or gray chalcedony and to satin spar (gypsum).
Oeil de boeuf	Labradorite (feldspar).
Oligoclase	Feldspar, silicate of aluminum sodium, and potassium.

Gem name	Common name or description
Opaline feldspar	Labradorite.
Orthoclase	Potash feldspar in monoclinic crystals, silicate of aluminum and potassium.
Orthose	Moonstone (feldspar).
Ox-eye	Labradorite (feldspar).
Peristerite	Iridescent albite (feldspar).
Perthite	Potash feldspar (orthoclase or microcline) with laminae of soda feldspar (albite).
Sunstone	Feldspar (usually oligoclase or labradorite) containing inclusions of minute scales of iron oxide.
Variolite	Dark green orthoclase (feldspar) containing lighter-colored globular particles.
Water opal	Moonstone (feldspar).
Wolf's eye	Moonstone (feldspar).

USES

Besides being used in jewelry, feldspars are cut into various shapes for personal adornment and for architectural decorations. Ordinary feldspar is used chiefly in the ceramic and glass industries, and to a minor extent as an abrasive in scouring preparations, but is also employed for a variety of other purposes.

IDENTIFICATION

The feldspars are readily distinguished by their optical properties, moderate hardness, pronounced cleavage, and peculiar color phenomena. Orthoclase and microcline are only distinguished from each other by careful examination. Both are difficultly fusible and insoluble in acids. They are distinguished from the other feldspars by their right-angle cleavage and the lack of striations on the best cleavage surface.

The plagioclase or soda-lime feldspars (albite-anorthite series) are a trifle more fusible and not quite so hard as the potash feldspars, but these varieties are so similar to one another that they can often be distinguished only by a chemical analysis or a thorough optical examination.

SUBSTITUTES

With the exception of microcline, anorthoclase, and hyalophane, all the feldspars have been synthetically prepared. Those requiring the use of fluxes for their preparation are orthoclase, celsian, and albite. Anorthite melts sharply at 1,550° C. However, there is little commercial advantage in manufacturing feldspar gems on account of their relatively low value and comparatively small use as semiprecious stones.

HISTORY

The name "feldspar" was used by Wallerins in his mineralogy in the Swedish

form feltspat, which means field spar. It was subsequently written "felspar" when it was erroneously assumed that the word was derived from fels, meaning a rock. Feldspar is derived from the German word feld, meaning a field, but in German the old name is preserved as "feldspat." The preferred American spelling is "feldspar," but in England "felspar" is in general usè. Both spellings, however, are commonly seen.

Adularia or moonstone

Gem varieties of orthoclase when opalescent are called moonstone, and adularia when transparent or colorless. The finest stones at one time came from the St. Gothard district in Switzerland and were known as adularia, from the neighboring Adular Mountains.

Amazon stone

Amazon stone, the opaque, green feldspar obtains its name from the Amazon River, where, however, none has ever been found. Beautiful green stones were found on the banks of the Amazon and these may have been confused with the mineral of the feldspar group or with jade or a similar mineral.

Labradorite

Originally the finest specimens of labradorite came from the Isle of St. Paul off the coast of Labrador, where they were first discovered in 1770. This stone gets its name from Labrador, which country still is its most important locality.

MODE OF OCCURRENCE

Feldspar occurs as an accessory constituent of all types of rocks; igneous, sedimentary, and metamorphic. Its most common occurrences are in granites, syenites, porphyries, certain sandstones and conglomerates, and in gneisses. Good crystals occur chiefly in pegmatite veins associated usually with quartz, muscovite, etc. Such veins are to be found in regions where granitic rocks abound.

MINING METHODS

Practically all the mining done in the United States for gem feldspar has consisted of pits, small open cuts, and occasional tunnels. The various workings for amazon stone and other feldspar minerals cover considerable ground where there are pits every few feet in most instances, but none of them are very deep. In fact, most of the workings are less than 12 feet deep.

DOMESTIC PRODUCTION

The production of feldspar gems in the United States has always been exceedingly small. Although the mineral itself is widely distributed through many States, feldspar of gem quality has not been mined to a considerable

extent, because only exceptionally fine stones are worth more than the cost of cutting. The following table gives the value of feldspar gems produced in the United States from 1900 to 1921, when the United States Geological Survey discontinued the canvass of producers:

Table 1. - Value of feldspar gems produced in the United States 1900-1921 1/

Year	Value	Year	Value	Year	Value
1900	$ 270	1908	$2,850	1915	$ 368
1901	200	1909	2,700	1916	305
1902	500	1910	2,510	1917	(2)
1903	400	1911	175	1918	(2)
1904	500	1912	1,310	1919	(2)
1905	1,000	1913	1,285	1920	520
1906	100	1914	449	1921	155
1907	1,110				

1/ Compiled from annual chapters of Mineral Resources of the United States, Part II.
2/ Less than three producers; figures not published separately.

DEPOSITS IN THE UNITED STATES

Although some gem orthoclase and oligoclase have been produced in the United States, and labradorite, sunstone, and moonstone have been found in certain localities, amazon stones seem to be more widely distributed and to have been produced to a greater extent than any of the feldspar gems.

Amazon Stone

California. - A number of specimens of amazon stone were collected in the deserts of California somewhere between Barstow and Lone Pine. This material consisted of fragments and crystals which ranged from small sizes to an inch in thickness. A few of the specimens sent to the United States Geological Survey were especially good shades of bluish green and greenish blue, with fairly smooth, fine texture. This locality should prove of value if larger crystals of the same good color can be obtained in quantity. However, this region has not been very extensively exploited up to the present time.

Colorado. - The Pikes Peak region has long been famous as a source of beautiful crystals of amazon stone and associated smoky and clear quartz. There are two localities where amazon stone is found, one to the east of Pikes Peak, in the Crystal Park region near Manitou Springs, and the other about 18 miles northwest of Pikes Peak, and about 4 miles north of Florissant. Some of the amazon stone is of good gem quality and has a rich green and bluish green color, while other stones are pale or badly stained with iron. These crystals of amazon stone are generally well developed, and vary in

form feltspat, which means field spar. It was subsequently written "felspar" when it was erroneously assumed that the word was derived from fels, meaning a rock. Feldspar is derived from the German word feld, meaning a field, but in German the old name is preserved as "feldspat." The preferred American spelling is "feldspar," but in England "felspar" is in general use. Both spellings, however, are commonly seen.

Adularia or moonstone

Gem varieties of orthoclase when opalescent are called moonstone, and adularia when transparent or colorless. The finest stones at one time came from the St. Gothard district in Switzerland and were known as adularia, from the neighboring Adular Mountains.

Amazon stone

Amazon stone, the opaque, green feldspar obtains its name from the Amazon River, where, however, none has ever been found. Beautiful green stones were found on the banks of the Amazon and these may have been confused with the mineral of the feldspar group or with jade or a similar mineral.

Labradorite

Originally the finest specimens of labradorite came from the Isle of St. Paul off the coast of Labrador, where they were first discovered in 1770. This stone gets its name from Labrador, which country still is its most important locality.

MODE OF OCCURRENCE

Feldspar occurs as an accessory constituent of all types of rocks; igneous, sedimentary, and metamorphic. Its most common occurrences are in granites, syenites, porphyries, certain sandstones and conglomerates, and in gneisses. Good crystals occur chiefly in pegmatite veins associated usually with quartz, muscovite, etc. Such veins are to be found in regions where granitic rocks abound.

MINING METHODS

Practically all the mining done in the United States for gem feldspar has consisted of pits, small open cuts, and occasional tunnels. The various workings for amazon stone and other feldspar minerals cover considerable ground where there are pits every few feet in most instances, but none of them are very deep. In fact, most of the workings are less than 12 feet deep.

DOMESTIC PRODUCTION

The production of feldspar gems in the United States has always been exceedingly small. Although the mineral itself is widely distributed through many States, feldspar of gem quality has not been mined to a considerable

extent, because only exceptionally fine stones are worth more than the cost of cutting. The following table gives the value of feldspar gems produced in the United States from 1900 to 1921, when the United States Geological Survey discontinued the canvass of producers:

Table 1. - Value of feldspar gems produced in the United States 1900-1921 1/

Year	Value	Year	Value	Year	Value
1900	$ 270	1908	$2,850	1915	$ 368
1901	200	1909	2,700	1916	305
1902	500	1910	2,510	1917	(2)
1903	400	1911	175	1918	(2)
1904	500	1912	1,310	1919	(2)
1905	1,000	1913	1,285	1920	520
1906	100	1914	449	1921	155
1907	1,110				

1/ Compiled from annual chapters of Mineral Resources of the United States, Part II.
2/ Less than three producers; figures not published separately.

DEPOSITS IN THE UNITED STATES

Although some gem orthoclase and oligoclase have been produced in the United States, and labradorite, sunstone, and moonstone have been found in certain localities, amazon stones seem to be more widely distributed and to have been produced to a greater extent than any of the feldspar gems.

Amazon Stone

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size from a fraction of an inch to 3 or 4 inches square; the larger pieces furnish fine cabinet specimens for mineral collections. Whitman Cross[4] and W. F. Hillebrand have described the minerals of this region.

The bulk of the feldspar used as a precious stone has been produced by Colorado, and the amazon stone leads all the others in value. George F. Kunz[5] states that an exhibit of Colorado amazon stone at the World's Fair in Philadelphia in 1876 occasioned much surprise because of its beauty.

Maine. - Amazon stone was found along the coast of Maine during 1914. This discovery was made late in the fall, so that little prospecting was possible and only about a pound of crystals was obtained. The specimens submitted to the United States Geological Survey were rather small but would cut into gems of pleasing color. If larger masses of equally good quality could be obtained this deposit would be of interest to the New England semiprecious stone trade. However, this locality has not been seriously exploited up to the present.

New York. - Prof. Freeman F. Burr, of Barnard College, New York City, submitted to the United States Geological Survey several crystals of amazon stone which he had collected from the quarry at White Plains, N. Y. This quarry was then being worked for materials to be used in the construction of Kensico dam for the New York City water supply. One crystal from this quarry, measuring 7 by 7 by 5 inches was placed in the collection of Columbia University. Some of these crystals were fairly well developed, and the best specimen submitted to the United States Geological Survey showed a bright translucent bluish green color; this stone, however, could not be cut into a pure gem on account of the markings. Professor Burr[6] describes the occurrence of amazon stone at White Plains, N. Y., and states that a few of these stones have been cut as gems but none have been regularly placed on the market.

Pennsylvania. - An occurrence of amazon stone with sunstone was reported to the United States Geological Survey about 1908, at a locality 1 mile west of Media, Delaware County, Pa. These minerals were found loose in the soil, where they were brought up from pegmatite ledges by the action of frost.

Virginia. - Fine specimens and gem minerals have been obtained from the mica mines of the Amelia Courthouse region, Virginia. A description of this locality and its minerals has been given by W. F. Fontaine,[7] who mentions evidences of work by Indians or other persons. Amazon stone abounds at these mica mines where it occurs in a coarse pegmatite associated with other varieties of feldspar.

4/ Cross, Whitman, and Hillebrand, W. F., Minerals from Pikes Peak: Am. Jour. Sci., 3d ser., vol. 24, 1882, pp. 281-286.
5/ Kunz, G. F., Gems and precious stones of North America: New York, 1890, p. 165.
6/ Burr, F. F., Occurrence of Amazon Stone at North White Plains, New York: Sch. of Mines Quart., vol. 36, 1915, pp. 186-188.
7/ Fontaine, W. F., Notes on the Occurrence of Certain Minerals in Amelia County, Va.: Am. Jour. Sci., 3d ser., vol. 25, 1883, pp. 330-339.

According to Doctor Foshag,[8] the color of the amazon stone of Amelia, Va., is more pleasing even than that of Colorado. He further states that although this is massive and not in large and freely developed crystals as in Colorado, it possesses a pleasing and uniform shade that is better suited for cutting into beads and other ornamental forms.

Labradorite

Oregon. - About 1908, a report of the discovery of a new deposit of labradorite in southern Oregon was made by Maynard Bixby, of Salt Lake City, Utah, who stated that this mineral would yield handsome gem material. This labradorite ranges from colorless to a dark variety, showing fine red, salmon, and green tints.

Utah. - A new find of transparent pale yellow labradorite from Millard County, Utah, was reported by Kunz.[9] At this locality it was found occurring in scoria and is of interest as a gem, although it is not of great hardness. It is almost identical with the occurrence of the Altar Mountains location of Arizona, which was the first reported instance of a transparent labradorite.

Moonstone

Virginia. - Specimens of moonstones obtained from a mica mine were received from Hewlett, Va. This variety of orthoclase feldspar occurs in pockets in veinlets of partly kaolinized feldspar in a decomposed mica gneiss formation, which pockets range from the size of an egg to that of a cocoanut. The material examined was slightly yellowish, not the pure white of good moonstone, and although it displays a certain amount of the chatoyancy of moonstone, it was not so strong as the Ceylon moonstone.

Sunstone

Arizona. - Dr. H. P. Wightman of Globe, Ariz., submitted to the United States Geological Survey specimens of andesine feldspar, some of which showed the characters of sunstone. These specimens were collected by the Apache Indians from their reservation not far from Globe, and resemble the sunstone from Modoc County, Calif. Microscopic examination showed it to be andesine with a refractive index of 1.550. While one piece of this material was a pale yellowish color, another showed bright copper-colored reflections from inclusions along certain lines of crystallization parallel to one of the cleavages.

California. - Quite a quantity of sunstone from Modoc County, Calif., has been cut into gems, specimens of which were received by the United States Geological Survey from the Pacific Gem Co. of Los Angeles, Calif. These cut gems are very pretty and have been sold under the name of both goldstone and sunstone. This feldspar comes rather high in calcium, with many minute inclusions that reflect a bright coppery red light. In some specimens these

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8/ Foshag, Wm. F., Gems and Gem Minerals: Smithsonian Sci. Ser., vol. 3, pt. 2, 1929, pp. 262-267.

9/ Kunz, G. F., The production of Precious Stones for the Year 1915: Min. Ind., 1916, p. 611.

particles are almost submicroscopic but sufficiently abundant to impart a red color. However, the body of the feldspar material is colorless and clear.

FOREIGN LOCALITIES

Amazon Stone

Amazon stone or amazonite, occurs chiefly in the Ilmen Mountains, Russia, Siberia, and Scotland. This bright variety of microcline is a very popular shade for brooches and the like, and is often cut cabochon on account of its pleasing color.

Labradorite

The foreign deposits of labradorite are found mainly in Finland, Russia, and on the coast of Labrador. Labrador is the most important locality.

Moonstone

Ceylon is the principal source of moonstone. In fact, practically all the moonstones on the market now come from Ceylon, although at one time the finest came from the St. Gothard district in Switzerland. In recent years extensive deposits of superb stones with rich reflections have been obtained and sent to the gem markets all over the world, with the result that Ceylon is now noted as a producer of the finest moonstones obtainable.

Sunstones

Sunstones of very excellent quality are produced in many parts of the world. According to Doctor Foshag,10/ rarity gave sunstone considerable popularity at one time, but it lost that asset when a number of localities yielding abundant quantities appeared, especially at Tvedestrand in Norway.

10/ Foshag, Wm. F., Work cited, pp. 264-265.

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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING LAWS OF PALESTINE

BY

E. P. YOUNGMAN

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF PALESTINE[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of Palestine was prepared from Bentwich's Legislation of Palestine[3] and from a report made by H. Gordan Minnegerode, American vice consul, at Jerusalem, in reply to a questionnaire submitted by the United States Bureau of Mines and transmitted through the courtesy of the State Department.

INTRODUCTION

The Ottoman law, which was in force in Palestine before the British conquest of the country (1917-18), still forms the basis of the legal system. Nevertheless, with respect to mining, legislation has been considerably revised since the British occupation and mandate, some of the Ottoman laws having been either radically altered or wholly abolished, in order that the legal status of Palestine might be more in harmony with that of western States. Legal transition is still far from being complete; however, no further mining legislation is pending at the present time (March, 1931).

Revisions during the British military occupation (1917-20) took the form of proclamations of the commander in chief. After the establishment of the civil administration (1920) changes in the legal system were a decided departure from the Turkish laws. In 1922 the practice was begun of giving consecutive numbers and short titles to the ordinances.

The basic mining law is the mining ordinance of 1925, No. 19 (amended by mining ordinance No. 10 of 1926), an "ordinance to make provision regarding the winning of minerals, and the conditions under which minerals may be gotten, and to replace the existing Ottoman laws relating thereto." The chief regulatory provisions are embodied in regulations (A) and (B)--mining ordinance, 1925. This mining legislation of 1925 repealed the following laws:

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1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6534."

2 Rare metals and nonmetals division.

3 Bentwich, Norman DeMattos, Legislation of Palestine, 1918-1925: Vols. 1 and 2, Whitehead Morris (Ltd.), Alexandria, 1926.

1. Ottoman Law of Mines of 14th Sefer 1324 (March 26, 1322).

2. Ottoman Law of 24th Moharrem 1327 (August 27, 1325), concerning transport in mines, as amended by the Ottoman Provisional Law of the 16th Jamad el Akhar 1332 (April 29, 1330).

3. Ottoman Provisional Law, amending the law of mines dated 14th Moharrem 1332 (January 1, 1329).

4. Ottoman Provisional Law amending the law of mines dated 7th Rajab 1333 (March 8, 1331).

5. The Ottoman Land Code, article 107.

Although the mineral industry of Palestine is still problematical, the outlook is considered promising. Recent discoveries of mineral deposits, many of them unknown before the British mandate came into effect (1923), indicate the possibility that the country is rich in hidden minerals. Eight prospecting licenses have been granted by the Palestine Government to one explorer and nine to another for operations in more or less the same area. Spring and well water are abundant in the vicinity of the known deposits, labor is cheap and plentiful, and a good natural harbor is nearby. Moreover, the existence of ore deposits in neighboring territory (notably, copper and manganese along the Gulf of Suez) has been taken as a strong indication that similar deposits may exist in the territory of Palestine and Trans-Jordan.

RIGHTS OF FOREIGNERS

Americans, as well as all other foreigners, subject to the same terms as are British subjects or natives of Palestine, may obtain the right to explore for minerals and to own and operate mines, by reason of a provision in the mandate granted to Great Britain by the League of Nations in 1922 and made effective in 1923, which prohibits any form of discrimination in Palestine on the ground of nationality.

No specification is made in the law that any definite percentage of mining rights, or leases, shall be vested in British subjects or in nationals. Wide discretionary power is given mining authorities in granting mining privileges, but under the terms of the mandate foreign enterprises are accorded identically the same rights and privileges as are British subjects or natives of the country.

INCORPORATION

Incorporation as a general requirement is not imposed upon concession holders, but a concessionaire may be instructed to incorporate as a condition of his concession. A number of recently granted concessions were subject to special ordinances, which provided for incorporation and for the investment of a specific minimum of capital.[4]

4 Knabenshue, P., American consul general, Jerusalem, The Mining Laws in Palestine: Consular Rept., Mar. 25, 1931, State Department file 800.63/933 (No. 288). (Data furnished by H. Gordan Minnegerode, as stated in the Foreword.)

CLASSIFICATION OF MINERALS[5]

Minerals in Palestine, for the purpose of the mining laws, are defined as "all materials of economic value forming part of or derived naturally from the crust of the earth, including mineral oil,[6] pitch, asphalt, and natural gas, but not minerals in solution, or peat, trees, timber, and similar kinds of forest produce, or the materials defined in article 1 of the Ottoman Law of Quarries, dated July 2, 1901." In general, minerals are classed as non-precious and precious.

1. Nonprecious minerals include all minerals except precious minerals.

2. Precious minerals include precious stones, precious metals, and the ores of precious metals.

(a) Precious stones include diamonds, rubies, sapphires, emeralds, and such other stones as the High Commissioner may, by proclamation published in the Official Gazette, declare to be precious stones.

(b) Precious metals include platinum, gold, silver, and such other metals as the High Commissioner may, by proclamation published in the Official Gazette, declare to be precious metals.

(c) "No ore shall be deemed to be a precious mineral unless there be present in the ore such proportion of precious metal or metals as would enable the ore in bulk to be mined in payable quantities apart from the proceeds realized by the extraction of any other mineral substance."

OWNERSHIP

Land

No distinction is made between the ownership of the soil and that of the subsoil unless it is made in a legal agreement. The owner of the surface land, unless he be the holder himself of a right from the Government, may not prevent prospecting, exploring, or mining upon his land. The law (sec. 1, art. 6, min. ord., 1925) reads:

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5 Sec. (i) to (n), art. 2, min. ord., 1925.
6 "Oil" means mineral oil and includes natural gas. (Sec. (s), art. 2, min. ord., 1925.

No prospecting or exploration permit or prospecting or prospecting oil licence, mining right, mining or mining oil lease shall be granted with respect to land other than public lands except with the consent of the owner of such land if such land be not leased for a period exceeding three years, or with the consent of the owner and lessee of such land if the land be so leased; provided that if such consent be not given within one month of application therefor being made by the controller, and such owner or lessee is not the holder of or applicant for an exploration permit, or of a licence, right, or lease with respect to the land in question, then a permit, licence, right, or lease may be issued or granted in accordance with this ordinance upon the applicant's giving such security as the controller shall require for the payment of reasonable compensation for the damage that may be done in the course of the mining operations so authorized.

Sections 1 and 2, article 35, mining ordinance, 1925, provide:

If any part of the land with respect to which application for a mining right or mining or mining oil lease is made is mulk[7] land or true wakf,[8] such right or mining lease shall not be granted with respect to such part save with the consent of the owner in regard to mulk land, or the competent authority in regard to true wakf.

Provided that, if consent be withheld and if in the opinion of the High Commissioner it is in the public interest that mining should proceed, the High Commissioner may grant such right or lease as though consent had not been withheld. The person or authority whose consent is dispensed with shall have a right to compensation payable by the holder of the mining right or the mining lease and assessed on the basis of indemnity for the damage caused. The amount of such compensation shall, in default of agreement, be determined by arbitration in the manner prescribed. Provided further that four-fifths of the royalty, other than any additional royalty payable under the following section, with respect to the minerals won or raised from land being mulk or true wakf shall be paid to the owner of such land or the competent authority as the case may be.

Mining shall be considered to be a "public purpose" within the meaning of any law or ordinance relative to the expropriation of land for public purposes. (Art. 8, min. ord., 1925.) See, for further statements concerning the right of the High Commissioner to expropriate land for mining purposes, article 43, mining ordinance, 1925.

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7 & 8 For definition, see a following paragraph.

"Fee-simple" ownership of land in Palestine is not known. The law differentiates between "mulk" (freehold), "miri" (leasehold), "chiftlik" (State domain), and "wakf." Mulk, which includes all the urban land, and which can be disposed of as chattels, most nearly corresponds to what in the United States is known as freehold. Miri, which includes practically all rural land, is not subject to disposition by testament but devolves according to a special law. Chiftlik, or State domain, comprises a large part of the land in Palestine. The "fellahin" (land cultivators in possession of such land) are entitled to the rights of cultivation but not of ownership. Wakf is a grant or dedication of land in trust for a religious purpose.

Minerals

Minerals obtained by mining belong to the holder of the mining right or lease, subject to certain rents and royalties. Minerals raised or obtained in the course of prospecting or exploring are the property of the Government of Palestine.

If the holder of a prospecting license or of a prospecting oil license shall desire to dispose of any minerals raised or obtained in the course of prospecting, he shall make application to the Controller of Mines in the prescribed manner; if the controller is satisfied that the prospector has been conducting only such work as is reasonably necessary to enable him to test the mineral-bearing qualities of the land, he may authorize the applicant to retain and dispose of the minerals upon payment of the prescribed royalties. (Art. 25, min. ord., 1925.)

MINING AUTHORITIES

The British High Commissioner is given power to appoint an officer to be styled "controller of mines" and such other officers as may be necessary for carrying into effect the provisions of the law. The office of controller of mines may, if the High Commissioner so rules, be combined with another Government office. "Director" means the director of the department or other officer of the Government charged with the administration of the mining ordinances. (Art. 2 and 4, min. ord., 1925.)

The controller of mines grants prospecting and exploring permits and licenses and certificates of discovery. He has power to grant a permit or license for the full area requested, to grant it for a smaller area than that requested, or to refuse it outright, if in his judgment the applicant is not eligible. (Art. 9, 12, 16, 17, 21, and 26, min. ord., 1925.)

The High Commissioner, or an officer designated by him, grants mining rights or mining leases. He may refuse an application, or he may grant it for the full area or for a part of the area requested. (Art. 29, 31, 32, and 33, min. ord., 1925.)

The High Commissioner, by notice in the Official Gazette, may declare any area closed to prospecting or exploring for either a specified or an unlimited period. (Art. 5, min. ord., 1925.) The only restriction, seemingly, placed upon the mining authorities is that they shall not discriminate upon the grounds of nationality.

According to subsection (b), section 2, article 6, mining ordinance, 1925, in every prospecting or exploring permit, prospecting or prospecting oil license, mining right, mining or mining oil lease there is implied--

> A reservation to the High Commissioner of all his powers to enter into and upon, and to grant to any person or persons whomsoever liberty to enter into or upon, the said lands for all and every purpose other than those for which the permit, license, right, or lease has been granted, and particularly (and without in any way qualifying such general power or liberty) to make or lead on, over, or through the said lands, such roads, tramways, railways, telephone and telegraph lines and pipe lines as shall be considered necessary and expedient for any purpose, and to obtain from and out of the lands comprised in such permit, license, right, or lease, such stones, timber, or other materials as may be necessary or requisite for making, repairing, or maintaining such roads, tramways, railways, telephone and telegraph lines, electric cables and pipe lines, or for any other purpose, and to pass and repass over and along the said lands and such roads, tramways, railways, and pipe lines for all purposes as occasion shall require, and subject to such abatement of rent or payment of compensation as may, in default of agreement, be determined by arbitration in the manner prescribed.

APPEALS

> Any person aggrieved by the refusal of the controller to grant a permit or license or a renewal thereof or to consent to its transfer or by the decision of the controller to cancel a permit or license .... may in the manner prescribed appeal to the High Commissioner, whose decision shall be final.
>
> Whenever a right of appeal is provided, the person aggrieved by the decision of the controller may require the controller to state the grounds of the decision in writing and may lodge an appeal in writing with the High Commissioner within 15 days of being furnished with the grounds of the decision. A decision on appeal may be given on the documents or after hearing the appellant and other persons. The High Commissioner may refer the appeal for hearing or for report to the attorney general. (Art. 23, min. ord., 1925.)

The controller has jurisdiction in all suits concerning any rights claimed in, under, or with relation to permits, licenses, rights, or leases, or any advantage thereof or liability thereunder, other than the liability to any penalty or suits concerning any contract with respect to permits, licenses, rights, or leases. This jurisdiction extends to suits in which the litigants (or one of them) are holders of a permit, license, right, or lease, or are owners or occupiers of the land comprised thereunder. The decision of the controller need not be formally drawn up but must be recorded. The controller may, as the result of a suit, cause minerals to be summarily seized and delivered to the person entitled to them. Appeal from the controller's decision may be made to the supreme court.

Any litigant, instead of bringing suit before the controller, may take the case to any of the ordinary courts of the land; but the court may, upon the application of the defendant or defendants, direct that the suit be heard by the controller. (Art. 87 to 92, min. ord., 1925.)

PROSPECTING, EXPLORING, DISCOVERY, AND MINING

General

Prospecting includes all operations connected with the search for minerals and reasonably necessary to enable the prospector to test the mineral-bearing qualities of the land. (Art. 2, min. ord., 1925.)

Exploring means the conducting of a geological survey to ascertain the probable presence or absence of minerals or other information as to the nature and structure of the earth's crust and the distribution of surface and underground water. (Art. 2, min. ord., 1925.)

Mining includes all acts reasonably necessary for the working, winning, getting, and raising of minerals to the surface and transporting them for the purposes of refinery or storage. (Art. 2, min. ord., 1925.)

Rights issued under the Palestine Government are classed as prospecting permits, exploring permits, prospecting licenses, prospecting oil licenses, mining rights, mining leases, and mining oil leases, and others, each of which is discussed separately.

Prospecting and Exploring[9]

Prospecting Permits

A prospecting permit may be limited or unlimited with regard both to the area over which and the minerals for which the holder may prospect. It shall not confer any exclusive right upon the holder with respect either to the area or the minerals. No prospecting permit shall confer the right to prospect for oil.

9 Art. 9 to 27, min. ord., 1925; art. 1 to 28, reg.--min. ord., 1925.

Application.- The applicant, who must state his name and address within the territory of Palestine and that of his principal (if any), shall apply in person at the office of the Mines Branch. The applicant must state the area over which and the minerals for which he intends to prospect.

Qualifications.- A permit shall not be granted to: any person not able to read; any person under 18 years of age; any person that has been convicted of an offence against the mining ordinance or that has forfeited a previous right under a permit, license, right, or lease by reason of a breach of the terms or conditions of that right; or any person not able to prove financial competency. A sworn declaration as to age and nationality may be required.

Limitations.- Except with the special permission of the controller, a permit holder shall not dig trenches more than 2 meters deep or sink shafts more than 10 meters deep. He shall not drill or take any other steps having as an object or direct result the winning of minerals. A prospecting permit is not transferable.

Duration.- A prospecting permit is issued for one year but may be renewed, at the discretion of the controller, for an indefinite number of times, under prescribed conditions.

Exploration Permits

An exploring permit is limited as to the area to be covered. It carries with it the exclusive right to the holder thereof to explore the area comprised in the permit and (subject to certain conditions) preferential and exclusive right to obtain a prospecting or prospecting oil license over portions selected by him, which shall not exceed 4 per cent of the area explored in the case of oil, 1 per cent in the case of the nonprecious minerals other than oil, and one-half of 1 per cent in the case of the precious minerals. The total perimeter of the portions (which shall be rectangular in shape) shall not exceed twice the perimeter of a square of the same superficial area as the portions.

Application.- An application for an exploration permit is made in practically the same way as one for a prospecting permit. In addition to stating the area he wishes to explore, the applicant must state the names and the qualifications of the geologists and experts he is prepared to employ.

Qualifications.- An exploration permit may be granted to any person that is able to produce satisfactory evidence that he possesses or commands sufficient working capital and technical knowledge or assistance to insure proper and adequate exploration and that will agree to undertake a geological and mineral survey of the area to be explored.

Size of area.- The area comprised in an exploration permit shall not exceed 500 square kilometers.

Duration.- The term of an exploration permit shall not exceed two years.

Duties.- The holder of an exploration permit shall employ competent geological surveyors and other experts (approved by the controller), who shall make the surveys directed by the controller and furnish him with such reports, information, maps, plans, diagrams, specimens, analyses, et cetera, as he may request.

Prospecting Licenses
(Other than Oil Licenses)

A prospecting license is issued for a definite area, which the applicant must identify, and for certain minerals, which the applicant must name. A prospecting license confers upon the holder thereof the exclusive right to prospect the area mentioned in the license during a period not exceeding the maximum prescribed for such mineral or minerals (other than oil), as shall be stated in the license.

Application.- An application for a prospecting license must be made in person at the office of the Mines Branch.

Qualifications.- The qualifications for holders of prospecting licenses are the same as those for holders of permits; for only those holding prospecting or exploring permits may apply for licenses. (Art. 16 and 17, min. ord., 1925.)

Article 20 of the regulations--mining ordinance of 1925--says:

> If the whole or a part of such area (for which a prospecting license is sought) is owned or occupied by a private person, the applicant shall produce evidence to the satisfaction of the controller that the applicant either himself or through his agent has prospected the area or the part thereof in question for the mineral or minerals in question with the consent of the said owner or holder of surface rights, or that such area or part thereof has not been prospected and the consent of the owner or holder of surface rights in question or his representatives has been obtained by the applicant, as required by section 6 of the ordinance.
>
> If the applicant is not in a position to prove either of the alternatives set out in the preceding regulation, a prospecting or prospecting oil license shall not be issued in respect to the area privately owned or occupied unless and until it appears that the owner or holder of surface rights is not prospecting and is not prepared to prospect the area himself or by his agents in a manner approved by the controller, when a prospecting or prospecting oil license may be issued to the applicant on his giving the security required by section 6 of the ordinance.

In addition, no application shall be granted unless the applicant shall--

(a) Produce satisfactory evidence that the mineral or minerals for which application is made exist in the area applied for.

(b) Shall have beaconed such area in the prescribed manner prior to making his application.

Area.- The form of an area granted by a license shall, as far as possible, be rectangular, and the shorter sides shall not be less than one-third of the longer sides.

Except when the applicant is the holder of an exploration permit, the size of an area shall not exceed 20 hectares in the case of precious minerals or 50 hectares in the case of nonprecious minerals other than oil.

Duration.- A prospecting license shall be issued for not less than 1 year nor more than 5; if granted for less than 5 years, it may be renewed at the discretion of the controller to a maximum of 5 years.

Notifications and records.- A licensee shall notify the controller of the position of any proposed borehole or shaft and submit a plan thereof. He shall keep an accurate record of every borehole or shaft sunk and hold at the disposal of the controller representative samples of minerals taken from each.

Prospecting Oil Licenses

A prospecting oil license is granted for a definite, explored area, named in the license. It gives the holder thereof the exclusive right of prospecting within the prescribed area.

Application.- An application for a prospecting oil license is made in person by the applicant to the Mines Branch.

Qualifications.- The controller may grant a prospecting oil license to any person able to prove that a satisfactory geological survey of the area in question has been made by a holder of an exploration permit or otherwise. (Art. 21, min. ord., 1925.)

Area.- Except when the applicant is the holder of an exploration permit, the maximum area under a prospecting oil license shall be 2 square kilometers.

Duration.- A prospecting oil license is granted for a period of one year but is subject to renewal at the discretion of the controller.

Discovery

Discovery of minerals in apparently paying quantities entitles the discoverer to a certificate of discovery, which entitles him (and his successors and assigns) to a mining right or mining oil lease, but for which

he must make application within one year from the date of the certificate. If the holder of such a certificate satisfies the controller that he possesses or commands sufficient working capital and technical knowledge to insure adequate development of the mineral resources, he alone is entitled to a mining right or lease, unless he should voluntarily waive his right. (Art. 28 and 29, min. ord., 1925.)

Every discoverer is required to place a beacon at the place of discovery (for manner, see art. 29 and 30, reg. (A)--min. ord., 1925) and to notify the controller (for form of notification, see annex to Schedule 1, reg. (A)--min. ord., 1925).

Mining[10]

Mining Rights and Mining Leases

All mining privileges are granted by either "mining rights" or "mining leases." No mining right shall be granted save in respect to alluvial minerals (sec. 2, art. 37, min. ord., 1925). The controller decides whether the circumstances of a case require that mining shall be carried on by virtue of a mining right or a mining lease (sec. 2, art. 30, min. ord., 1925).

A mining right or a mining lease is granted only to a holder of a certificate of discovery for one year after the issuance of the certificate, unless the discoverer waives his right. Should the discoverer waive his right and another receive the mining right or lease, the High Commissioner may impose an additional royalty and decide what proportion thereof shall be paid to the discoverer.

Application.- An application for a mining right or lease by a discoverer of minerals shall be made personally or by a representative; application by any one other than a discoverer shall be in writing.

Qualifications.- Applicants for mining rights or leases, whether holders of certificates of discovery or not, may be required to prove that they possess or command sufficient working capital and technical skill for the operations contemplated.

No person under the age of 21 years shall hold directly or indirectly a mining right or lease, except in a representative capacity. (Art. 56, min. ord., 1925.)

Mining Rights[11]

Size of area.- The area covered by a mining right (granted for alluvial minerals only) shall not exceed 1 square kilometer, or in the case of valley alluvium 1 kilometer along the course of the stream and 100 meters on either side from the center of the stream.

10 Art. 30 to 41, min. ord., 1925; art. 31 to 35, reg. (A)--min. ord., 1925.
11 Art. 37, min. ord., 1925; art. 31 to 35, reg. (A)--min. ord., 1925.

Duration.- A mining right may be granted for one year and may be renewed by the High Commissioner for successive periods of one year.

Rights obtained.- A mining right gives the exclusive right to mine within the area granted and to take and dispose of any mineral obtained, upon the payment of the prescribed rent and royalties.

Mining Leases[12] (Other than Oil Leases)

Size of area.- The total area for which a mining lease may be granted to any one company or may be held by a group of associated companies shall not exceed twice the maximum area obtainable under an exploration permit.

The area covered by a mining lease shall not exceed the maximum over which a prospecting license may be granted with respect to the mineral to be mined, except with the consent of the High Commissioner:

Provided that in no other case shall the unit leased be greater than 20 hectares where precious metals are concerned or 50 hectares where nonprecious metals are concerned.

Duration.- A mining lease shall not be granted for a period exceeding 30 years. As long as minerals are produced in payable quantities, the lessee shall have preferential right to a renewal.

Rights obtained.- Upon the payment of the prescribed rent and royalties, the lessee may remove and dispose of the minerals specified in the lease. For a definite statement of the usual rights of a mining lessee, see articles 38 and 39 of the mining ordinance of 1925.

Mining Oil Leases[13]

Size of area.- The total area for which leases shall be granted to any one company or shall be held by a group of associated companies shall not exceed twice the maximum area obtainable under an exploration permit. (See also art. 38, min. ord., 1925.)

The area of a mining lease shall not exceed the maximum area over which a prospecting license may be granted in respect to the mineral to be mined, except with the consent of the High Commissioner: Provided that in no other case shall the unit leased be greater than 100 hectares with respect to oil.

Duration.- An oil mining lease shall not be granted for a period exceeding 30 years. As long as oil may be produced in paying quantities, the lessee shall have the preferential right to a renewal of his lease.

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12 Art. 38 and 39, min. ord., 1925; art. 31 to 35, reg. (A)--min. ord., 1925.
13 Art. 40 and 41, min. ord., 1925; art. 31 to 35, reg. (A)--min. ord., 1925.

Rights.- A mining oil lessee has the right to enter upon the lands subject to his lease and--

(a) To bore, dig, sink, drive, make, repair and use all such boreholes, pits, shafts, drifts, levels, excavations, waterways, and other works as may be necessary or proper for the purpose of searching for and winning oil, subject to such regulations and general directions limiting the number and depth of oil and gas wells as may be prescribed from time to time with a view to due conservation and control of the oil resources of the country.

(b) Subject to the provisions concerning water rights (see section of this paper entitled "Miscellaneous"), to appropriate and use for any purpose connected with the borings or works referred to in the preceding paragraph the water under any of the said lands that may be made available by such operations and to bore for such water and to collect and impound same for the purpose of such operations.

(c) To enter upon, use, and occupy a sufficient part of the said lands adjoining any borings for depositing thereon the products of the said borings and all the earth, soil, and other substances brought to the surface and for otherwise carrying on the works of the said borings.

(d) To refine oil in and upon the said lands whether for the purpose of sale or otherwise, save as hereinafter provided.

(e) To store, take, lead, pipe, and carry away, on, under, or over the said land the crude oil and refined products and to dispose of the same, subject to such conditions as may be prescribed.

(f) To erect, set up, and make in, upon, and over the said lands houses for his agents, workmen, and other employees, sheds, engines, machinery, furnaces, buildings, erections, pipe lines, tramroads, and other roads, and works necessary and convenient for the effectual working of the said borings, diggings, or works connected therewith.

Surface Leases[14]

The holder of a mining or mining oil lease, for his mining enterprise, may obtain the "lease of the surface of an area in excess of the area comprised in his lease" by applying to the High Commissioner and by paying the prescribed fees. The lease shall be effective for a period equal to the unexpired portion of his mining lease, and it may be renewed with the mining lease. The powers of the High Commissioner to expropriate the land necessary are set forth in detail in article 43 of the mining ordinance of 1925.

14 Art. 42 and 43, min. ord., 1925.

AREAS EXEMPTED FROM EXPLORATION, PROSPECTING, AND MINING

By section (2), article 5, mining ordinance of 1925, the holders of permits, licenses, rights, or leases may not operate within:

(a) (i) Any area that comprises or is within 100 meters of a holy site, except with the consent of the religious body intrusted with its control.

(ii) Any area that comprises or is within 100 meters of an historical site, except with the consent of the director of antiquities.

(iii) Any closed forest area, State or State-controlled forest, except with the consent of the director of agriculture of forests, and subject to such conditions as may be imposed by him for the protection of forest produce.

(b) Land situate within a municipal area or area occupied by a village, except with the consent of the municipal or local authority.

(c) Land reserved for the purpose of any railway or situate within 100 meters of any railway, except with the consent of the general manager of the railways.

(d) Land the site of, or within 100 meters of, any dam or reservoir used for the purpose of supplying water to the public, or land the site of, or within 100 meters of, any pipe line used in connection with such dam or reservoir, except with the consent of the public authority concerned.

(e) Land which the High Commissioner may by notice declare to be closed either temporarily or permanently to exploration, prospecting or mining.

(f) Land over which an exploration permit, a prospecting or prospecting oil license, or mining right previously granted shall be still in force, except by, or on behalf of, the person to whom such permit, license, or right has been granted.

(g) Land held under valid mining leases and land with respect to which application for a mining right or lease has been made until such application has been refused.

(h) Land over which any right of prospecting or mining has been granted or acquired under the late Ottoman Government, unless and until the High Commissioner shall by notice declare such land to be open to exploration or prospecting.

The authorities may make an exception with respect to lands covered by paragraphs (f), (g), and (h) by granting the privileges of exploring, prospecting, or mining for minerals other than those covered by existing permits, if the interests of the holders of the latter are not prejudiced thereby. (Sec. 3, art. 1, min. ord., 1925.)

DAMAGES[15]

The holding of a prospecting or exploring permit, prospecting or prospecting oil license, mining right, mining or mining oil lease implies an obligation upon the part of the holder to compensate the owner or the occupier of the land for any damage caused by exploring, prospecting, or mining to any surface rights (created before or after the grant of the permit, license, right, or lease). If the controller shall decide that damage is likely to be caused, he may require the applicant to give security against damage (either by deposit or otherwise) before the issuance of the permit or other document. If such security is given, the owner or occupier of the land may not restrain operations.

TRANSFER

All mining interests (including the rights conferred by any permit, license, right, or lease) shall be capable of being transferred (whether absolutely or by way of charge), or surrendered, or amalgamated, or otherwise dealt with in the same manner as other interests in the land, provided that every transfer shall have the approval of the director, and provided that every transfer or surrender shall be published in the Official Gazette. (Art. 73 and 74, min. ord., 1925.)

The holder of a mining right or mining or surface lease, in order to surrender his right or lease, must gain the consent of the High Commissioner; but surrender shall not relieve the holder of a right or lease from performing the duties imposed upon him and due to be performed at or before the date of surrender. (Art. 50, min. ord., 1925.)

ABANDONMENT

The holder of a prospecting or exploration permit or of a prospecting or prospecting oil license may, by giving notice, abandon his prospecting or exploration area. He must, upon abandonment, fence or fill in all shafts, pits, holes, and excavations. (Art. 26, min. ord., 1925.)

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15 Art. 6, min. ord., 1925.

FORFEITURE

Failure on the part of the holder to exercise skill and diligence may result in the cancellation of an exploring permit (art. 14, min. ord., 1925), of a prospecting license (art. 18, min. ord., 1925), or of a prospecting oil license (art. 22, min. ord., 1925).

A mining right or lease may be forfeited by order of the High Commissioner, under the following conditions:

1. If the rent or the royalty remains unpaid for six months after the date upon which it is due.

2. If a breach has occurred in any of the covenants of the lease (except those relating to rent, royalties, and labor conditions), and if the breach continues after the director has given written notice concerning the delinquency.

3. If the holder of a mining right or lease is guilty of wasteful practices in connection with the operation of the mine.

The High Commissioner is authorized to use his discretion in extending the period during which the defaulting holder of a mining right or lease may comply with the conditions of the right or lease and to specify whatever conditions he may consider just.

The High Commissioner must insert in the Official Gazette a notice of every forfeiture of a mining right or lease, Such a notice is evidence that the land and the minerals may be dealt with as though no right or lease had every been granted. (Art. 51, min. ord., 1925.)

FEES, RENTS, AND ROYALTIES[16]

Fees

	Egyptian pound[17]
1. Issue of a prospecting permit	2
2. Renewal of a prospecting permit	1
3. Issue or surrender of an exploration permit	5
4. Issue, renewal, or surrender of a prospecting license	5
5. Fee payable per annum per 10 hectares or part thereof by holder of prospecting license	1
6. Issue, renewal, or surrender of a prospecting oil license	5
7. Fee payable per annum per 10 hectares or part thereof by the holder of a prospecting oil license	1
8. Issue of a mining right	2
9. Renewal of a mining right	1
10. Grant or surrender of a mining lease	25

16 Pt. 1, 2, and 3, Schedule 1, min. ord., 1925.
17 The exchange value of the Egyptian pound in 1929 was $4.98.

Fees - Continued	Egyptian pound
11. Grant or surrender of a mining oil lease	25
12. Issue, renewal, or surrender of a surface lease granted under the provisions of section 42 of the ordinance	10
13. Issue, renewal, surrender, or enlargement of a water right	10
14. Prior to the hearing of an objection to the grant of a water right, under the provisions of section 67 of the ordinance, the fee of LE.5 for each day of 5 hours for which such hearing lasts with a minimum fee of LE.5; such fee to be paid in the first instance by the objector or the person showing cause, respectively, and to be deemed costs in the cause	
15. Trader's license per annum , ...	10
or if granted after June 30 in any year	5
16. For permission to lessee to dispose of minerals other than those specified in the lease	2
17. For permission to the holder of a prospecting permit, prospecting license, prospecting oil license, or exploration permit to remove, retain, or dispose of minerals	2
18. On withdrawing an application for an exploration permit or prospecting oil license, mining right, mining or mining oil lease	5

	Turkish piaster[18]
19. Registration of any document (other than a prospecting or exploration permit, prospecting or prospecting oil licence, mining right, or mining or mining oil lease)	50
20. For search in register for every hour or part thereof	25
21. For copy of, or extract of, any registered document, per every 100 words	10

18 The value of the Turkish piaster in 1928 was 4.4 cents in U. S. currency.

Rent

	£m/ms[19]
1. Holder of a mining right-- Rent payable per 100 meters along the stream per annum	5.000
2. Lessee of a mining lease--	
(a) In the case of a lease giving the right to work precious minerals, for every hectare or part thereof	5.000
(b) In the case of a lease giving the right to work nonprecious metallic minerals, for every hectare or part thereof	5.000
(c) In the case of a lease giving the right to work nonprecious nonmetallic minerals, for every hectare or part thereof	2.500
3. Lessee of a mining oil lease-- In the case of a lease giving the right to work mineral oil, for every hectare or part thereof	2.500

Royalties

Royalties on Oil and Natural Gas

(Royalty payable by the lessee of a mining oil lease or of a mining lease of bituminous limestones or shale for the purpose of extracting oil.)

1. A royalty per ton of the net crude oil received into the lessee's field storage tanks or reservoirs during the preceding 12 months at the following rate, that is to say:

	Per cent
On the first 10,000 tons	5
On each ton exceeding 10,000 but not exceeding 50,000	$7\frac{1}{2}$
On each ton exceeding 50,000 but not exceeding 90,000	10
On each ton exceeding 90,000 but not exceeding 130,000	$12\frac{1}{2}$
On each ton exceeding 130,000	15

Provided always that in cases where leases are obtained by public tender the royalty shall only be limited by the highest tender received.

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19 The Standard of currency from November 1, 1927, is the Palestine pound (£.P.), divided into 1,000 mills and equivalent in value to the pound sterling. The Statesman's Year-Book, 1930, The Macmillan Co., New York, p. 192.

For the purpose of these regulations net crude oil means crude oil after deducting water and all foreign substances and includes any casing-head gasoline obtained. If the crude oil upon which royalty is payable is of more than one grade, the lessor shall be entitled to receive such proportion of each grade as the total amount of that grade bears to the whole of the oil upon which royalty is payable.

2. A royalty of 1 piaster per 1,000 cubic feet of natural gas sold by the lessee and calculated at an absolute pressure of 1 atmosphere and at a temperature of 60° F.

3. At the option of the lessor or lessee the royalties prescribed by the two preceding regulations may be paid half yearly on January 1 and the July 1 in respect to the amount of net crude oil received or natural gas sold by the lessee during the preceding six months, but so that in the result the amount of royalty paid shall not be less than that hereinbefore provided, having regard to the total amount of net crude oil received by the lessee in any one year.

4. The lessor may in respect to crude oil at his option take a royalty in kind or may require the lessee to pay its equivalent money value; but unless the lessor shall on or before March 31 in any year by notice in writing to the lessee require the lessee to pay the royalty in kind, it shall be deemed that the lessee shall be bound to pay the equivalent value of the royalty in money as hereinafter provided.

5. The equivalent money value above mentioned shall be fixed on January 1 of each year by agreement between the lessor and lessee, and, failing agreement, by arbitration as prescribed by the ordinance; and in assessing such value regard shall be had to the average value during the previous year of each grade of crude oil taken at the field storage tank, such value to be calculated, where quotations for crude oil in the open market are not available, by taking the prices obtainable for the refined products and deducting the reasonable cost of refining, transport, and marketing, plus a profit of 10 per cent, on these operations; no other deductions being allowed.

Provided that during the first two years of the lease the royalty per ton of net crude oil may, at the discretion of the lessor, be fixed at P.T. 20 per ton without prejudice to the power of the lessor to take royalty in kind during these years.

Provided further that the said rate of P.T.20 per ton having been fixed arbitrarily in order to secure the advantage of a fixed rate during the said years, no inference shall be drawn from the amount of the said rate in respect to the real value of the crude oil or products thereof, either before the arbitrators referred to in this regulation or otherwise.

6. Should the lessee have erected a refinery for the treatment of crude oil, the lessor may, if he has given notice as provided in this regulation of his intention to take royalty in kind, take in lieu thereof quantities of refined products equivalent to the amount of crude oil payable by way of royalty in kind, due allowance being made for the cost of refining.

7. If the lessor has given notice in writing of his intention to take all or part of the royalty in kind, the lessee shall, within three months of the date from which the payment of royalty becomes due and on payment of all proper charges for the transport and storage of the oil, provide all reasonable facilities at his disposal to enable the lessor to take delivery in kind at any point in the territory at his discretion, whether in the form of crude oil or refined products.

8. For the purpose of the above regulations, when a company holds more than one mineral oil lease in Palestine, the royalties referred to above shall be assessed on the total production of the company from all leases held by it, and when a company owns a controlling interest in one or more subsidiary companies, the total production of the controlling and subsidiary companies shall be aggregated for this purpose.

The controlling interest for the purpose of this regulation means not less than the holding of 75 per cent of the issued share capital for the time being of the subsidiary company.

Royalties on Other Minerals

Precious minerals.- 5 per cent of the product or the equivalent value thereof at the mine.

Nonprecious minerals.- 2 per cent of the product or the equivalent value thereof at the mine.

The equivalent value shall be fixed in the way prescribed in regulation 5.

Royalties shall be payable to the controller half-yearly, in arrears. Rents with respect to mining rights and leases shall be paid to the controller annually, in advance. Surface rents, under-surface leases, shall be payable half-yearly, in arrears. (Art. 47, min. ord., 1925.)

A penalty of 5 per cent is placed upon failure to pay rent (or any other sum payable under a mining right or mining lease) before or on the date of payment; and a penalty of an additional 10 per cent is placed upon the sum due, exclusive of the 5 per cent penalty, if not paid within one month of the date for payment. In case of further delinquency, the director may recover the sum due by action in court. (Art. 49, min. ord., 1925.)

MISCELLANEOUS

Trader's license.- No person shall purchase, trade in, or receive precious minerals unless he be the holder of a trader's license, in the form prescribed. The director may, at his discretion, grant or refuse a trader's license. Such a license expires on December 31 of the year in which it is granted; it is not transferable.

The holder of a trader's license shall keep (and submit to the director upon request) records of all transactions effected under the license. He shall satisfy himself that the seller is authorized to be in possession of such minerals as he seeks to dispose of before purchasing, trading in, or receiving them. The licensee shall not pay any person in his employ by means of the precious minerals, except in the form of coins issued as currency. (Art. 94, min. ord., 1925.)

Registration.- Every prospecting or exploring permit, prospecting or prospecting oil license, mining right, mining or mining oil or surface lease, or water right granted, as well as every instrument by or under which such privileges or any part thereof shall be transferred, surrendered, amalgamated, or otherwise dealt with, shall be registered with the controller; every such permit, license, right, lease, or certificate shall be null and void if not presented for registration within one month of the date thereof or of the date of an extension of time.

The registrant, upon paying the prescribed fee, will receive a certificate of registration.

Every instrument required to be registered in the Mines Branch must be registered with the Land Registry also. (Art. 75, min. ord., 1925.)

Extralateral rights.- Article 7 of the mining ordinance of 1925 provides that--

> The rights to minerals in respect to any area of land shall be limited to the minerals within the vertical boundaries of such area and shall not extend to any continuation of mineral lodes, veins, reefs, or beds beyond such boundaries, provided that the right to take mineral oil, pitch, or asphalt shall not be affected by the fact that such material flows into the area from outside its limits, and provided that no borehole for mineral oil or natural gas shall be drilled within 30 meters of the boundaries of the surface area leased.

Water rights.- No proprietary right to water or the right to divert water is comprised in any permit, license, right, or lease. Details with respect to the obtaining of water rights and the form of the application therefor are found in articles 64 to 72 of the mining ordinance of 1925 and in articles 38 to 40 of regulations (B)--mining ordinance of 1925.

Regulatory provisions.- The purely regulatory provisions of mining, covered in regulations (A) and (B) of the mining ordinance of 1925, deal with the following subjects: Demarcating, marking, beaconing, and surveying of areas; water rights; forms of permits, licenses, leases, rights, and certificates; surface-land protection; control and inspection of mines; ways, works, and machinery; ventilation; sanitation; lighting; explosives, drilling, and blasting; employed persons, accidents; plans; and mineral returns.

Detailed provisions covering operations in oil fields are given in articles 78 to 118 of regulations (B) of the mining ordinance of 1925.

I. C. 6535 November, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING LAWS OF AUSTRIA

BY

E. P. Youngman

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF AUSTRIA[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of Austria was prepared chiefly from reports made by American representatives[3] at Vienna and transmitted to the United States Bureau of Mines through the courtesy of the Department of State and of the Bureau of Foreign and Domestic Commerce.

INTRODUCTION

Prior to 1854 the provincial legislatures in Austria enacted their own mining laws. The confusion that resulted from the existence of different codes when mining began to be conducted by modern methods forced the drafting of a law to cover all mining operations of the country. Such a general law (allegemines österreichisches Berggesetz) came into effect May 23, 1854, in the old monarchy, with the exception of Hungary, which had its own special laws. This general law of 1854 is still the basic mining law for Austria, although it has been amended and supplemented by a number of acts.

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1 - The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6535."
2 - Rare metals and nonmetals division, U. S. Bureau of Mines.
3 - Richardson, Gardner, commercial attaché, Austrian Mining Law: Spec. Rept. 32, Feb. 12, 1931 (made in answer to a questionnaire of the U.S. Bureau of Mines and approved by an official of the Mining Department, Ministry of Commerce and Communications, Austria).
Groves, H. Lawrence, commercial attaché, Mining Laws of Austria: Spec. Rept. 26, Nov. 9, 1928 (made in answer to a questionnaire of the U. S. Bureau of Mines and approved by an official of the Austrian Mining Department, Dr. Streinz).
Heingartner, Robert H., consul, Austrian Mining Laws: Consular Rept. 394, Dec. 19, 1923 (approved by officials of the Ministry of Commerce and Communications).
Groves, H. Lawrence, commercial attaché, Austrian Law Favors Oil Exploitation: Econ. and Trade Notes 68, Mar. 1, 1929.
Perkins, C. Warwick, vice consul, Prospective Drilling for Oil and Natural Gas in Austria: Consular Rept., Dec. 7, 1928 (approved by Councillor Franz Aggermann, Chief of the Mining Department, and by others).

Hungarian legislation still applies to the Province of Burgenland, Austria. However, no essential difference exists between the mining law of Austria and that of Hungary, as Hungary in 1868 prescribed for itself the full text and decisions of the General Austrian Mining Law.

The basic law and its amendments are as follows:

General Austrian Mining Law (allgemeines österreichisches Berggesetz), May 23, 1854, State Legislative Bulletin 146 (R. G. Bl. Nr. 146)

Mineral oil and mineral gas law (Erdöl- und Erdgasgesetz), July 7, 1922, Federal Legislative Bulletin 446 (B. G. Bl. Nr. 446)

Law governing drilling discoveries (Bohrfundgesetz), Sept. 26, 1923, Federal Legislative Bulletin 535 (B. G. Bl. Nr. 535)

Mining wage law (Bergbaulohnzahlungsgesetz), May 17, 1912, Federal Legislative Bulletin 107 (R. G. Bl. Nr. 107)

General law governing the simplification of Federal administration (allgemeines Verwaltungsentlastungsgesetz), July 21, 1925, Federal Legislative Bulletin 277 (B. G. Bl. Nr. 277)

Law governing the fees for free prospects (Freischürfgebührengesetz), May 12, 1925, Federal Legislative Bulletin 160 (B. G. Bl. Nr. 160)

The following are the more important of the supplementary laws:

Law governing the establishment and province of the mining authorities (Gesetz über die Einrichtung und den Wirkungskreis der Bergbehörden), July 21, 1871, State Legislative Bulletin 77 (R. G. Bl. Nr. 77)

Law governing operating instructions for free prospecting (Bundesgesetz betreffend die Erlassung vom Betriebsvorschriften für Freischürfe), Oct. 20, 1921, Federal Legislative Bulletin 587 (B. G. Bl. Nr. 587)

Law governing the establishment of a social welface fund for miners (Bundesgesetz betreffend die Errichtung eines Bergbaufürsorgefonds), Dec. 3, 1925, Federal Legislative Bulletin 432 (B. G. Bl. Nr. 432)

General mining police regulations (allgemeine Bergpolizeiverordung), Aug. 26, 1928, Federal Legislative Bulletin 238 (B. G. Bl. Nr. 238)

Law governing the employment of women and children and the working hours in mines (Bergarbeitergesetz), July 28, 1919, Federal Legislative Bulletin 406 (S. G. Bl. Nr. 406)

The mineral wealth of Austria has not been exhausted. The following table, giving production figures for 1929, is a fair index of the importance of Austria's mineral industry. Petroleum, no production of which has been recorded, is discussed in a separate section of this paper.

Mineral production in Austria in 1929[1]

Mineral	Metric tons	Value (schillings)[2]
Antimony ore	11,290.7	240,000
Barite	300.1	7,606
Briquettes	420.0	18,192
Cement	450,000.0	- - -
Coal (bituminous)	208,020.0	6,514,502
Lignite	3,524,792.0	65,145,966
Coke (gas-house)	62,894.8	- - -
Copper ore	135,113.5	2,706,092
Graphite (crude)	25,296.1	854,753
Gypsum	43,000.0	- - -
Ichthyolic shale	670.4	52,962
Iron (manganese content)	1,891,381.0	15,567,433
Pig iron	458,973.0	56,857,729
Kaolin	33,050.0	- - -
Lead and zinc ore	115,024.9	2,729,571
Lead-smelter product	6,569.0	5,207,241
Magnesite (crude)	438,000.0	- - -
Mineral paints:		
Ochre	1,071.8	20,432
Iron oxide	1,136.2	178,624
Metallic quicksilver	3.9	81,442
Rock salt	3,040.5	- - -
Salt brine	[3]5,551,311.0	- - -
Gold-silver (bullion?)	.3	50,556
Talc	17,000.0	- - -

1/ Harris, Ernest L., American consul general, Vienna; Consular Rept., Aug. 11, 1930, Bureau of Mines foreign file 2919.

2/ The 1930 exchange value of the Austrian schilling was 14.0891 cents in U. S. currency.

3/ Hectoliters.

CLASSIFICATION

With the exception of salt, which as a State monopoly is in a class by itself, minerals fall into but two classes: (1) "Reserved" minerals (vorbehaltene Mineralien), the disposal of which belongs exclusively to the Government; and (2) "all others," which belong to the owner of the surface.

The so-called "reserved" minerals include all those that can be utilized because of their content of metal, sulphur, alum, or sodium chloride; waters that are of value "because of their metal admixture"; graphite, asphalt, mineral oil, mineral gas, all minerals that are "technically valuable because of their bitumen content," and all varieties of pit coal and lignite.

OWNERSHIP

The General Austrian Mining Law reiterated and made binding a vital principle (expressed in earlier laws) from the point of view of national economy--that is, that the exploitation of mineral resources should be independent of and above property rights. A certain group of minerals, therefore, was reserved to the "exclusive right of disposal by the sovereign" (now the State). Such reserved minerals may be prospected for and mined only by the express authorization of the Government, in accordance with the provisions of the law. The "sovereign right" by which certain minerals belong to the sovereign (State) is called Bergregal.

In order to prospect on his own land for the "reserved" minerals, the owner must obtain permission from the Government; but otherwise the owner of the surface land has the right of ownership to the air column above, to the surface land, and to the subsoil.

RIGHTS OF FOREIGNERS

Any foreigner is on equal terms with nationals with respect to mining operations in Austria, provided that the country of his origin grants like privileges to Austrian citizens. This principle of reciprocity is enunciated in the Civil Code (bürgerliches Gesetzbuch).

A nonresident foreigner must have a legal representative within Austria; likewise, partnerships or companies must have a joint representative in the country. Foreign stock companies require an "admission," in accordance with an imperial decree of November 29, 1865 (State Legislative Bulletin 127). Incorporation of a foreign company under the laws of Austria is not required. (See also section of this paper entitled "Mining Companies.")

Americans and other foreigners are not discriminated against either by law or by factors outside of the law; on the contrary, foreign initiative in the mining field is encouraged by the Austrian Government, which is anxious further to develop Austria's natural resources. Labor laws, affecting foreigners and nationals alike, are more exacting and burdensome to the mining industry than they are to other industries. At this time the heavy expenses in connection with old-age insurance (approximately 5 per cent of the pay roll) must be borne by the mining operators, as the special funds (Bruderladen) from which this insurance was paid formerly were lost through the depreciation of the currency. The total cost to the employer for welfare purposes has been estimated at 25 per cent of the average pay roll. (See also section of this paper entitled "Workmen's Insurance.")

Neither the law nor the Government specifies that a certain percentage of ownership in mining enterprises must be in the hands of nationals.

MINING AUTHORITIES

The mining authority of the Austrian Republic is the Mining Department, Ministry of Commerce and Communications;[4] in some circumstances the Ministry

4 - Groves, H. Lawrence, and Heingartner, Robert H. Works cited.

of Finance serves as a final court of appeal.[5] Seven district mining boards (Revierbergämter) form subordinate administrative authorities, located at St. Pölten, for Lower Austria; Wels, for Upper Austria and Salzburg; Leoben and Graz, for Styria; Klagenfurt, for Carinthia; Hall, for Tirol and Vorarlberg; and Sauerbrunn, for Burgenland.

A Revierbergamt grants both prospecting and mining rights in its district. It has no right to refuse a permit or a renewal thereof, provided the fees are paid and the prescribed amount of work is done. As this provision of the law is much criticised by local experts, a change is contemplated.

Mining matters are subject to certain mining courts of first instance (Berggerichte), composed of special mining senates, which consist of judges and technically trained advisers. These courts are located in the Federal States: at St. Pölten for Lower Austria, at Steyr for Upper Austria, at Salzburg for the State of Salzburg; at Leoben and Graz for Styria; at Klagenfurt for Carinthia; and at Innsbruck for Tirol and Vorarlberg.

PROSPECTING PERMITS

No prospecting may be done with respect to the "reserved" minerals by any one (not even the surface owner) without a Government permit. A prospecting permit does not specify a certain mineral or minerals but covers all "reserved" minerals.

The general permit (allgemeine Schurfbewilligung) does not grant an exclusive right, as more than one permit may be issued for the same area. An exclusive right may be obtained by the registration of what is termed "Freischurf," or "free prospect," which is the right to exclude any other prospector, irrespective of prior rights, within a circular area having a radius of 425 meters. Registration consists in notifying the mining authorities of the location of the central point chosen for operations. The notification must contain a reference to a prospecting license already granted, or it must be accompanied by an application for such a license. It must be accompanied also by documentary proof that the annual fee of 5 Austrian schillings has been paid. A Freischurf, covering the same period for which the prospecting permit is issued, and applying to unlimited depth and height, is effective on the day upon which the registration reaches the authorities.

<u>Application</u>.--An application for a prospecting permit is made to the Revierbergamt of the district in which the prospecting is to be done.

<u>Qualifications</u>.--Except that an applicant for a permit must be of age (21 years old) and must possess the usual qualifications for the free exercise of civil rights, no conditions are imposed by the law.

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5 - Richardson, Gardner, Work cited.

Duration.--A prospecting license is issued for one year, but it may be extended from year to year.

Area.--The area covered by an ordinary prospecting permit is not definitely limited; that covered by a Freischurf has a diameter of 850 meters.

Work required.--Although the Austrian law specifies that a certain amount of work must be done to keep a permit in force, in view of the fact that the requirements are exacting, the decision, in each case, of the amount of work to be done has been left to the supervising authority (Revierbergamt). The Austrian Government intends to modify the requirements and then to inforce its rulings strictly, thereby limiting the discretionary power of the local supervisors.

Disposal of minerals.--Neither a general prospecting permit nor a Freischurf gives the holder an ownership claim upon the minerals within his prospecting area or the right to mine them. Prospectors desiring to acquire ownership of the "reserved" minerals must apply to the proper authority for a concession or a grant. A prospector has the right to demand that the ownership of a limited quantity (two or three carloads) of minerals found by him in a deposit be given to him by the local Revierbergamt. A prospector disposing of "reserved" minerals for which a grant has not been given, without a provisional permission from the mining authorities, is liable to a penalty amounting to the equivalent of the minerals used or disposed of.

Exemptions.--Without the consent of the owner of the land, prospecting is not permitted within dwelling houses or farm or other buildings; in back gardens, flower gardens, or other fenced-in gardens; or in cemeteries or walled-in fields. Without the consent of the administrative authorities, prospecting is not permitted on public thoroughfares, railway tracks, waterworks, protective structures, or the frontiers of the Republic.

MINING

A "Verleihung des Grubenmasses" gives not only mining rights but also the full property of the deposit in question. This right, which is not a lease, concession, or grant (in the general meaning of such terms), but full ownership, is given after an official inspection has proved that the deposit in question justifies exploitation and is given upon the condition that mining actually will be undertaken.

Application.--An application for a mining right (made to the local Revierbergamt) must be accompanied by a detailed map, showing the exact location of the deposit.

Priority right.--The law gives the holder of a Freischurf the right to demand mining privileges over one or several units in the area allotted to him for prospecting purposes, but it does not give him priority as to choice

of the size or the location of the units. In actual practice, a prospector that locates deposits of minable minerals always succeeds in putting in the first claim for mining and ownership rights over worthwhile parcels.

Duration.--A mining right, being a property right, is unlimited as to time.

Areas.--The unit of the mining area is the "Grubenmass," which is rectangular in form, which extends to an unlimited depth and height (ewige Höhe und ewige Teufe), and which measures 45,116 square meters. The shorter side of the rectangle must not be less than 106 meters. (Fractions, or "Überscharen," are those parts that are so surrounded by allocated Grubenmasse that a regular unit can not be laid into them.)

As a rule, only one Grubenmass is granted in an open lode. However, four double Grubenmasse may be given in the case of pit coal and lignite or four simple Grubenmasse in the case of any of the other "reserved" minerals. The Revierbergämter may, and frequently have, granted more than the stipulated number of units when local conditions warranted their doing so. With respect to minerals found on the surface in natural deposits or on old abandoned burrows, areas as large as 115,000 square meters have been allowed.

A placer unit is termed "Tagmass." Tagmasse are granted for "reserved" minerals appearing in sand banks, in river beds, in surface deposits, in rocks formed by sediments, in old abandoned pit heaps, and in "bean" and bog ores. The Tagmass does not, as a rule, extend beyond the rock itself.

Auxiliary rights.--"Hilfsbaue," or shafts and by-pits, beyond the limits of the allocated field, may be constructed and operated only with the permission of the mining authorities.

"Revierstollen," or adit levels, "mining enterprises whereby an entire mining district is opened by shafts or whereby operations are facilitated," may be constructed and operated only with the permission of the mining authorities.

Obligations and rights.--The chief obligation of the holder of a mining privilege in one or more units is that of actually mining without interruption. Another obligation called a "construction-guarantee duty" (Bauhafthaltungspflicht) is that of safeguarding persons and property. Non-compliance with this duty is punishable by a withdrawal of mining rights. Another duty of the mine owner is that of keeping correct mine charts and furnishing statistical data to the mining authorities.

The usual mine owner's rights (based upon the peculiarity and the economic importance of the mining industry) include those of erecting the necessary buildings, foundries, mills, water-power plants, and water-supply systems; building roads, railways, and bridges, as well as other means of communication; and practicing all the handicrafts necessary to mining operations in special workshops, without the necessity of obtaining a separate and special concession therefor.

TRANSFERS

Both prospecting permits and mining rights may be transferred to other persons. No special permission is required, but a notification of every transfer must be sent to the Revierbergamt. A Freischurf may be transferred only to a person already holding a prospecting license or applying therefor at the time of the transfer.

CANCELLATION AND FORFEITURE

Public interest may at times justify the cancellation or the modification of either a prospecting right (Freischurf) or a mining right. In actual practice cancellation has been confined almost entirely to the withdrawal of oil rights when the owners therof declined to undertake exploration or exploitation and the Government transferred the rights to a group willing to undertake drilling.

Of course, mining rights may be cancelled by way of punishment for the nonfulfillment of obligations on the part of the owner of a right; but cancellation is exceptional and does not take place in a properly operated mine.

A penalty (500 schillings for the first offense and 1,000 schillings for a repitition) is imposed for failure to comply with the police regulations or to observe the safety measures prescribed by the mining authorities. If safety rules are disregarded in spite of repeated warnings or penalties, the authorities are entitled to withdraw the license.

RIGHTS AND OBLIGATIONS OF LANDOWNERS

Every surface owner has the right to inspect every prospecting permit covering his land and must be given a chance to come to an amicable agreement with a permittee concerning compensation. But the landowner may not prevent prospecting; for if no agreement is reached, the permit holder is authorized to apply to the authorities for permission to commence prospecting and at the same time to request that the authorities institute proceedings against the landowner to compel surrender of the land. According to the general mining law, a landowner is obliged to cede, for appropriate remuneration, the parcels of ground required by a prospector, provided he can prove that actual prospecting took place during the preceding year.

TAXES AND FEES

The regular industrial taxes are levied upon mining enterprises.

A fee of 5 schillings is collected for each permit.

Each mining unit ("Mass") is subject to a fee (payable semiannually)--usually about 10 schillings a year for pit coal and lignite deposits and about 5 schillings a year for other "reserved" minerals.

MISCELLANEOUS

Mining Companies

Mines may be operated by several co-owners or by companies, such as share-holding companies. A special form that is often encountered in Austria is the "Gewerkschaft," or mining association, which is distinguished from a share-holding company by the fact that the associates, if necessary, may be forced to give special assistance, which can not be claimed from shareholders. An associate, however, is responsible only to the value of his share of the assets of the Gewerkschaft. The shares in mining associations are termed "Kuxe." Not more than 128 Kuxe may be issued, and a Kux may not be divided into more than 100 parts, Thus the capital of an association may not be subdivided into more than 12,800 shares, except under exceptional circumstances and with the sanction of the authorities.

Monopolies

Monopolistic concessions are not granted in Austria.

Salt is under a State monopoly, according to the customs and State monopoly act of July 11, 1835, whereby all salt, whether in springs or in subterranean rock, is subject to a State monopoly; therefore, no mining claims for salt may be granted. The salt mines of the State are subject to the provisions of the general mining law, and the mines and the plants are under the supervision of the mining authorities. Surface owners' rights are greater in salt mining than in other mining, especially with respect to the expropriation of land and buildings.

Mining Registers

At each mining court a mining register is kept in which mining rights are inscribed, with the same legal effect as entries in the land registers.

Workmen's Insurance

By virtue of a law of July 28, 1889, mine employees are insured against old age or invalidity, through the "Bruderladen." Moreover, sickness insurance in industrial undertakings applies to the mining industry. Insurance for the members of miners' families also exists. Upon the depreciation of Austria's currency and the subsequent depreciation of the Crown sick fund, contributions were increased by legal measures, a law of April 16, 1920, providing that insurance rates be increased by supplements. These supplements, which the employer must pay, amount to 5 per cent of wages. The largest amount of insurance a miner was entitled to receive before the World War was 240 crowns ($48). In December, 1923, the highest amount was 3,600,000 crowns ($51) for one man, 1,800,000 crowns for a widow, 950,000 crowns for an orphan--the amount of orphan money being limited so as to keep the total amount of insurance paid to a widow within the 3,600,000 crowns allowed each miner. An orphan losing both parents was receiving 1,200,000 crowns annually.

The Bruderladen no longer insure against accidents; therefore, miners are insured with the general insurance companies. Accident insurance for miners was introduced in 1919.

Labor Conditions

Children under 14 years of age may not be employed in mines. Juvenile miners under 18 years of age may be employed only in work that will not impair their physical development. Female workers may be employed only above ground. Female workers of any age or juvenile male workers may not be employed at night--that is, between 8.00 p.m. and 5.00 a.m.

Working Hours

Working hours are regulated by a law of July 28, 1919. The 8-hour shift is the rule. Under certain special contracts, however, a longer daily shift may be arranged for on the basis of a 48-hour week; and in continuous operations the weekly hours may exceed 48, provided the total number of hours does not exceed 168 within three weeks. For underground operations working hours are calculated surface-to-surface.

PETROLEUM

Austrian imports of crude oil and petroleum products amounted in 1927 to 183,194 metric tons, valued at $6,767,000, and were increasing at the rate of 15 to 20 per cent annually. In order to reduce this factor in an already highly unfavorable balance of trade and in order to encourage the investment of foreign as well as domestic capital in the exploitation of oil-bearing areas in Austria, the Austrian Parliament passed an act on January 17, 1929, which provided that companies boring for oil or gas between September 1, 1928, and December 31, 1931, should be exempted from the corporation tax (Körperschaftssteuer), under the following conditions:

> That the boring and construction work be conducted by Austrian firms and that the machinery and other equipment be of domestic origin--the use of machines and other equipment of foreign origin not to be excluded when it is not available within the country or, if available, only at much higher prices or in much poorer quality.
>
> That after the expiration of two years from the date of finding oil the crude petroleum be distilled in plants within the country, in the form of either semimanufactured or finished products, such as gasoline, kerosene, benzene, gas oil, paraffin oil, lubricating oil, residues, and crack distillates. (The mere "taking off" of benzene and kerosene or the application of a less thorough distillation process than that indicated will not be considered as a fulfillment of the conditions for tax exemption.)

That the exemption from taxation be for a maximum period of six years from the first discovery of oil on a commercial scale by the company in question.

Petroleum Areas

Geological surveys and test borings made in various Austrian districts in the last two decades indicate that the Vienna Basin may be oil-bearing. This basin, which is about 125 miles long and 40 miles wide, belongs to the Alp-Carpathian Mountain complex. Two-thirds of it is in Austria, and one-third is in Czechoslovakia; it extends in a northeast-southwest direction from Göding (Czechoslovakia) to Wiener-Neustadt (Lower Austria) and in an east-west direction from the Leitha Mountains to the so-called "Vienna Forest" (Wienerwald). At least three well-developed dome structures exist: (1) near Zistersdorf, on the slopes of the Steinberg, north of the Danube; (2) at Lanzendorf, south of the Danube, about 13 kilometers southeast of Vienna; and (3) at Schwadorf, also south of the Danube, and about 32 kilometers southeast of Vienna. A comparatively recent report of these areas is as follows:[7]

> The most northern dome is that on the slopes of the Steinberg, where the Steinberg Naftagesellschaft (allied with the French Limanowa concern) and the firm of Raky have put down drillings. The Nafta Co.'s shallow hand borings, carried to a depth of 40 to 60 meters, yielded traces of gas and oil and indicated the axis of the anticline and the site for the first deep well, which is now in progress. Raky had already commenced a deep drilling a little over a mile southwest of the Nafta well, but work has been temporarily suspended at 250 meters, apparently both for financial reasons and with the object of awaiting the outcome of the Nafta operations.
>
> The Lanzendorf and Schwadorf domes are south of the Danube. Drilling, undertaken more than 10 years ago to locate lignite deposits, encountered strong traces of oil and gas at about 600 meters and showed that the summit of a large dome (Lanzendorf) was about a mile away. Geophysical tests located the dome accurately, and the Wiener Erdöl A.-G. is now drilling a well, which has touched 700 meters; the Sarmatian formation, where the oil horizons are, is expected at 800 meters. In view of the proximity of the City of Vienna, with its millions of inhabitants, it is surprising that more than 10 years elapsed before the discovery of gas was followed by serious operations. The adjoining Schwadorf dome also is in the hands of the Wiener Erdöl Co. Further promising dome structures have been located at Moosbrunn, Wolkersdorf, and Paasdorf; and others are probable in the Marchfeld area.

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7 - Friedl, Dr. Karl, Is Oil Likely to be Found Commercially in Austria?: Pet. Times, vol. 22, No. 560, London, Oct. 5, 1929, pp. 650 and 659.

Southeast of the Vienna territory lies the Hungaro-Croatian late Tertiary basin, of which only marginal portions are within the Austrian border; in this region indications of oil have been found in Bergenland and South Styria. In the Austrian Alps, oil is known to occur in many localities, but only at two, Wels and Leoprechting near Taufkirchen, has serious drilling been carried out. At Wels (Upper Austria) gas was discovered in 1891, and 150 boreholes, varying in depth from 150 to 350 meters, have supplied altogether 90,000,000 cubic feet of gas. In 1902-03, a deep drilling was carried to a depth of 1,044 meters, but with unsatisfactory results, perhaps owing to its not being placed close to the center of the dome. At 1,037 meters the granite underlying the basin was struck. At Leoprechting a highly viscous oil was encountered; borings in 1925 and 1926 yielded 150 tons. As the oil was too thick to be exploited by wells, a shaft was sunk to work the sand, which is on the average 8 meters in section, and which contains 40 per cent of oil. Operations, however, were abandoned for legal reasons, and there has been no activity since.

Further dome structures in Upper Austria have been proved at Grieskirchen, Henhart, Taiskirchen, Vöcklabruck, Bachmaning, and southwest of the famous spa Bad Hall.

A symposium, describing conditions with respect to petroleum in the Vienna Basin (both in Austria and in its extension into Czechoslovakia) recently appeared.[8]

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8 - Petroleum: Vol. 27, No. 6, Berlin, Feb. 4, 1931, pp. 91-105.

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Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING LAWS OF ETHIOPIA (ABYSSINIA)

BY

E. P. Youngman

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF ETHIOPIA (ABYSSINIA)[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the rights of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the laws of Ethiopia was prepared from the best available information in Washington and was checked against the answers made by Addison E. Southard, American Minister and Consul General, at Addis Ababa, to a questionnaire of the United States Bureau of Mines, which was transmitted through the courtesy of the State Department.

SOURCES OF LEGAL INFORMATION

In the writing of this report access was had to the following legal documents: Imperial Decree of the Ethiopian Government, April 8, 1928 (Miazia 1, 1920); Imperial Decree, December 5, 1929 (Year V, No. 49); official notice, March 19, 1931 (Yekatit 26, 1923--Year VII, No. 13); and official notice, May 21, 1931 (Genbot 10, 1923--Year VII, No. 21).

According to a recent report,[3] a new set of mining laws and regulations has been drafted, but considerable delay is anticipated before this new legislation will be officially approved.

INTRODUCTION

Ethiopia is generally conceded to be rich in minerals. The lack of mineral development in that country is attributable not so much to the inaccessibility of the deposits as to political conditions. Ethiopia is the only country in the Continent of Africa that has maintained its political independence; and now that it has a progressive and enlightened ruler, Ras Taffari, hope is entertained in many quarters that through foreign participation the Ethiopian mining industry will develop rapidly.

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1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from the U. S. Bureau of Mines Information Circular 6536."

2 Rare metals and nonmetals division, U. S. Bureau of Mines.

3 Southard, Addison E., American Minister and Consul General, Addis Ababa, Ethiopian Mining Laws: Consular Rept., Bureau of Mines foreign file 71, State Department file 24540, June 2, 1931.

Political Conditions

Although Taffari (son of Ras Makonnen and grandnephew of Menelik) was not crowned Emperor until October, 1930, as Regent of all Ethiopia and as King of Shoa (the chief Province), he has long been the virtual ruler. Still under 40 years of age, the new Emperor has set himself the task of improving conditions. Already (March 19, 1931), an official notice has been promulgated that sets aside the authority of the feudal barons and makes legal only the mining permits and mining concessions granted by the Imperial Government. This notice is interpreted as an important move of the central Government to establish its power over all divisions of the Empire; and there is no indication that it can long be opposed. The barons of Ethiopia, who govern the provincial territories, live as did the feudal barons of Europe 1,000 years ago, and hitherto they have maintained the attitude that the wealth of the country is theirs and should not be shared with foreigners. These barons as a class oppose the Emperor's progressive policies; therefore the central Government may encounter difficulty in protecting foreigners and in doing away with past practices, whereby bribery was rampant and obstacles (mainly indirect ones such as progressively increasing taxes or labor outbursts) were put in the path of foreigners attempting to carry on mining operations.

Among other political factors of possible influence upon the mining situation in Ethiopia are the activities of the three European powers holding adjacent territory. Under the terms of the treaty of December 13, 1906, the United Kingdom, France, and Italy agreed individually not to act in any way that would be injurious to one another with respect to Ethiopian affairs. Among the provisions of the treaty there is one concerning the construction of railroads that tends to exclude participation by other countries.

No expenditures in mining enterprises in Ethiopia should be made by a foreigner until he has a written agreement worked out by direct negotiations with the Emperor, or until he has consulted his own Government's representative in the Empire. In spite of the decree of April 8, 1928, mining rights are still a matter of personal negotiation; grants may or may not conform to that decree. The Emperor is still, in fact, the Department of Mines.

Economic Conditions
(Transportation)

In Ethiopia, a country of about 400,000 square miles of mountainous, rough land (called the "Switzerland of Africa"), transportation may afford one of the chief physical problems to be met by prospective mine operators and investors. Most of the prospects are remote from the capital, and the only railroad is the one that connects the capital (Addis Ababa) with Djibouti, a port in French Somaliland. Even this railway lies to the east of the capital, whereas the mineral wealth of the country apparently lies mainly west, southwest, or northwest.

From the gold and platinum concessions in the Province of Beni Shangul (which lies along the Sudan frontier), however, metals could be easily and economically shipped to Kurmuk and then transported by truck and rail through the Sudan to Khartum or Port Sudan. Platinum and gold, in fact, could be carried to Khartum by airplane.[4]

Through a treaty, August 2, 1928, between the Ethiopian and the Italian Governments, Ethiopia agreed to build a highway from Addis Ababa to the frontier of Italian Eritrea (probably to be built by Italian interests to the account of Ethiopia). That road would greatly increase the profitableness of the deposits of saltpeter in the Province of Sellale, especially those near Fiche, Debra Libanos, and Derra, which are 51, 66, and 84 miles, respectively, from Addis Ababa.[5]

A motor road could be built, it is said,[6] to the coast from a large sulphur deposit on the boundary line between Ethiopia and Eritrea. However, the difficulty of obtaining native labor and European supervisors would be very great, as the district has one of the most trying climates in the world, for it lies in hot, dry, desert country, 15 miles from water.

Mineral Production

Owing to the fact that the newly established Department of Mines lacks sufficient trained personnel, no Ethiopian Government statistics of mineral production are compiled. According to local exporters, the principal mining activity in 1929 was in gold and platinum. Gold mining was partly organized, but platinum production was the result of only scattered, individual operations. In 1929 about 5,000 ounces of gold probably was produced, of which the Government received 2,000 ounces from the Province of Beni Shangul, 2,000 from the Wollaga district, and 250 from Gubba. About 250 ounces appeared in the Addis Ababa market, for sale chiefly to foreigners. About 500 ounces was produced in other parts of Ethiopia. With respect to platinum, a local buyer reported a production of 240 kilograms in 1929. The only other mineral production for 1929, according to an American representative in Ethiopia,[7] was 10,000 metric tons of rock salt.

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4 Furness, James W., Exploitations of Mineral Areas in Ethiopia: Comm. Repts., No. 40, Oct. 7, 1929, p. 30-31.

5 Furness, James W., Work cited.

6 Robertson, H. P., Travelling in Abyssinia: Min. Mag., vol. 40, No. 6, London, June, 1929, p. 335-339.

7 Park, James L., American vice consul, Addis Ababa: Consular Rept., Apr. 23, 1930, Bureau of Mines foreign file 8955.

Mineral Wealth

The following miscellaneous statements concerning mineral deposits in Ethiopia are taken from annual reports made to the United States Bureau of Mines by American diplomatic representatives at Addis Ababa.

Asbestos.- Asbestos is known to exist, but commercial deposits have not yet been located.

Coal.- Large deposits of coal of good quality exist in the Ankober River area, of which Ankober (100 miles northeast of Addis Ababa) is the center. The deposit is about 70 miles from the railroad.

Copper.- Large deposits of copper occur in several localities in Ethiopia. One promising, high-grade occurrence is in the Ankober district. Another rich deposit lies in the Harrar district, in a highly mineralized contact zone, running roughly east and west from Gildessa to the Char Char Mountain. Sufficient prospecting and assaying have not yet been done to determine the real value of these occurrences. The most promising occurrences are in the southwestern parts, in the Omo River district. They are described as enormous and of high grade, extending over a highly mineralized area of more than 7,000 square miles. Plans were made in 1929 to develop copper in the Maji district (southeastern section of Ethiopia), but the effort is expected to fail until roads make export practicable or until better smelting equipment is installed.

Gold.- Gold occurs in many parts of the country and is generally washed out from river sands by natives, working individually. The Blue Nile, the Dabus and its tributaries, the Beni Shangul, and the Tumat Rivers have been mentioned. Quartz gold also is found, in the western and central parts. A survey of gold occurrences was made in 1926 in several provinces, such as Gojjam, Gurage, Shoa, and Wollaga, but the results were kept secret.

Iron.- Iron occurs in different parts of Ethiopia, but as far as is known not in such quantities as to warrant its development. In the Province of Tigre the natives smelt the richer ores over wood fires.

Limestone.- Limestone occurs in many sections, and a large dolomite formation is found in the Harrar district.

Mica.- Mica is plentiful in many parts of the country. The only reported production, however, is that of 2,000 kilograms in 1927 in the Harrar district. Marketable mica has been reported[8] in the Wollaga Province, about 250 miles west of Addis Ababa; in the Guinir district, 200 miles southwest of Dere Daoua; in the Aossa district, 150 miles north of Dere Daoua; and an American company at one time exploited a mica deposit 50 miles south of Dere Daoua and shipped about 6 tons of mica to London, New York, and Prague. This company was forced to cease operations because of management and other difficulties peculiar to Ethiopia.

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8 Robertson, H. P., Work cited.

Petroleum.- There was some agitation in 1920 and 1921 over petroleum concessions, and some drilling for petroleum has been done. However, no production has been reported. Petroleum indications have been observed, and one company, the Asiatic Co., a subsidiary of the Royal Dutch Shell group, has done some drilling. There is some belief in a great dome of oil under the Province of Harrar, as well as under British Somaliland. Several oil companies are now engaged in efforts to obtain a monopoly of the sale of petroleum products in the country.

Phosphate rock.- A large deposit of phosphate rock was discovered by a German prospector about 28 miles southeast of Harrar, in the Arussi country. It is said to be like the deposits of Egypt, occurring in a very dry district.

Platinum.- The platinum output is believed to come from ancient gold-mine dumps, worked about 1,000 B. C. It, as is the gold, is washed from sand by natives. One concession granted to an Italian company is being exploited by modern methods. Platinum was reported in 1926 to have been found on the banks of the Didessa River.

Potash.- An apparently inexhaustible deposit of potash, occurring in the Assalt salt plain, near the border of Eritrea, northwest of Lake Alelbad, was worked during the World War but now has a restricted production. The deposit is said to rival in size that of Stassfurt, Germany, but to lack the great variety of salts found in the German deposit. An Italian company is working some large nitrate-of-potash deposits profitably.

Rock salt.- The mining of rock salt is decreasing because of the imports of salt from French and British Somalilands. One very large deposit occurs in the Assali plain, northwest of Lake Alelbad. (This deposit is discolored with iron.)

Silver.- Silver is said to be found in the district south of the Baro River, in southwestern Ethiopia. Traces of silver have been found in the form of silver lead, about 25 miles south of Dire Dawa, in the Harrar district, within 8 miles of the railway. The deposit, which is extensive, consists of galena veins several inches thick, but of unknown depth, in the black limestone.

Sulphur.- Immense deposits of sulphur occur in the Harrar district, in the region of the Hawash River. Although not thoroughly investigated, it has been declared to be one of the most wonderful in the world, consisting of three rounded, cone-shaped mountains, 2,000 feet high, of nearly pure sulphur. The deposit lies within 20 miles of the railway.

Tin.- Several veins of tin occur in granite rocks in the Harrar district. Alluvial tin, in encouraging quantities, has been found in streams draining the Char Char Mountains. The largest deposits are in the Dire Dawa River, below Harrar.

Tungsten.- Tungsten is reported to exist, in interesting quantities, in the western section. It is reported in association with tin in the Harrar district. After an experimental exportation of Ethiopian tungsten (reported to be satisfactory) in 1928, no further production seems to have been undertaken.

The adjoining map (fig. 1) indicates the location of the chief mineral deposits, as well as the route of the one railway (between Addis Ababa and the coast) and the territories surrounding Ethiopia.[9]

Recently Reported Concessions

A French company (Société Minière des Concessions Prasso en Abyssine), with a capital of 9,000,000 francs, is working alluvial platinum deposits at Jubdo, on the Birbir River, near the head waters of the Blue Nile, about 16 days' trek on mule back west of Addis Ababa.[10]

Other recent concessions, as reported by Furness,[11] are as follows:

The Deutsches Studiensyndikat, of Stuttgart, Germany, holds a concession from the Emperor for the exploitation of all mineral deposits in 100 areas, each 5 by 5 kilometers square, or about 900 square miles, in the Province of Beni Shangul. Gold and platinum are the principal minerals, which are mostly in the river beds. Work was to begin in the late fall of 1929.

An Italian company has obtained a concession in the Provinces of Beni Shangul and Wollaga, where rich alluvial deposits of gold and platinum exist. The concession consists of seven circles, each having a radius of 5 kilometers. Several of the areas have been worked, and platinum was sold therefrom in Europe in 1928. A dispute between the concessionaire and the Government has been placed in the hands of arbitrators in Addis Ababa.

A saltpeter concession granted to a citizen of Ethiopia is north of Addis Ababa, in the districts of Borana, Jamma, Gundubret, and others, in the Province of Sellale. In the raw state the product contains 70 per cent of potassium nitrate, which increases to 95 to 97½ per cent by recrystallization, Similar deposits have been found at Debra Libanos, Harro Fitche, and Derra, the most important being at Derra, district of Jamma. Copper and coal have been found in these districts but have not been exploited, and alluvial gold is known to exist. The terms of the concession allow the holder to select seven different sections for exploitation, each of which is to have an area of 9 square miles. It will be difficult to exploit the minerals in this concession profitably until the highway between Addis Ababa and the Eritrean frontier shall have been built.

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9 Robertson, H. P., Work cited.
10 Robertson, H. P., Work cited.
11 Furness, James W., Work cited.

Figure I.- Location of the chief mineral deposits in Ethiopia (Abyssinia)

RIGHTS OF FOREIGNERS

Foreigners may not, as a rule, hold land in Ethiopia; but Americans, as well as other foreigners, are in theory and apparently in law permitted to explore for minerals and to own and operate mines in Ethiopia on equal terms with citizens of the country. No restriction so far has been placed upon capital by determining that a certain percentage of ownership in mines shall lie with Ethiopian nationals.

All corporations must register with the Ministry of Commerce, but no legal provision requiring incorporation under local laws exists at the present time.

Although American citizens do not enjoy the extraterritorial rights in Ethiopia that they do in many other countries, and American consular jurisdiction does not extend to an American citizen that has violated Ethiopian law (American law applying only when the violation is considered a crime rather than a breach of regulations), still an American consul may sit with the Ethiopian judge; and if he disagrees with the decision of the judge, he may appeal to the Emperor, whose decision is final. Of course, the interests of an American citizen may be further protected through diplomatic representations if the decision of the Emperor is considered a denial of justice.

OWNERSHIP

All mineral substances, except construction materials, now belong to the State. The decree of April 8, 1928, reads:

> All the wealth of the Ethiopian subsoil is national property, and the right of disposal, therefore, is withdrawn from the landowner. (Art. 1, decree of Apr. 8, 1928.)
>
> From the point of view of the application of the present decree, the following are considered as mines: Natural deposits of mineral substances or fossils, susceptible of a special utilization. Materials of construction, which remain free at the disposition of the landowner, are excepted. (Art. 2, decree of Apr. 8, 1928.)

No precedent indicates that fee-simple ownership is possible. Ethiopian land laws are practically nonexistent, and hence questions are likely to be decided arbitrarily on the basis of immediate expediency.

For the purpose of conserving the mineral wealth of the Empire of Ethiopia and of guaranteeing the control of its exportation, a decree was passed December 5, 1929,[12] with special reference to "platiniferous and

12 Decree of the Ministry of the Interior, in the name of His Majesty, King Taffari, Regent Plenipotentiary and Heir to the Throne of the Empire of Ethiopia, from Berhanena Salam, Addis Ababa, Year V, No. 49, dated Dec. 5, 1929.

auriferous substances, in the form of metals and ores," and "gems and precious stones," whereby "prospecting, searching, excavating, extracting, or reaping in subterranean as well as on surface exploitation," without an authorization by the Department of Mines of the Imperial Government, is subject to a fine of 500 to 2,000 thalers (Maria Theresa thalers)[13] and to the confiscation of the products won, through the Ministry of the Interior. A later notice, dated May 21, 1931, which is not retroactive, expressly reserves to the Government "all mines located in river beds and lakes" and prohibits all prospecting and mining therein by natives or foreigners (except the extracting of gold "by peasants," under the authority of the Department of Mines).

For the control of exportation, the decree of December 5, 1929, requires an export customs declaration and a certificate of authorization (from the Bureau of Mines), the concordance of which with the shipment must be verified by the customs officials, who are required to stamp the authorization delivered by the Bureau of Mines to indicate its cancellation whenever shipment shall have been made. A violation of these regulations is subject to a maximum fine of 1,500 Maria Theresa thalers and the confiscation of the products. Repeated violation within a period of one year will result in a doubling of the maximum fine.

Any person reporting and assisting in the prosecution and conviction of violaters will be entitled to a reward of one-third of the fines imposed.

AUTHORITY

Although the Department of Mines is the legally recognized authority in mining matters, and Dr. Robert Hesse (a German mining engineer) is the titular head thereof (Conseiller du Gouvernement Imperial Ethiopien Department des Mines), most matters are handled directly by the Emperor himself, who usually makes the final decisions concerning the granting of a lease or a concession, and who is not necessarily influenced by the opinion of other officials. He generally discusses matters directly with the lessee or the concessionaire.

A recent notice (dated March 19, 1931) is evidence of the determination of the Emperor to make the power of the central Government supreme with respect to the issuing of all permits. Certain feudal chieftains have not fully recognized this power and have issued mining permits or concessions in their own territory, a practice that has caused confusion and loss of money, especially to foreigners. The notice is as follows:[14]

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13 The Maria Theresa thaler, or dollar, is worth currently 49 cents.
14 Official notice, Yekatit 26, 1923 (Mar. 5, 1931).

The Department of Mines, by order of His Majesty the Emperor, informs interested parties that permits for prospecting and for the operation of mines in all Ethiopian territory shall be granted only by the Department of Mines of the Imperial Government of Ethiopia.

Any contract for a mining concession signed by an Ethiopian department other than the Department of Mines, at Addis Ababa, or by a provincial governor or chief, or by a landowner, shall be considered null and void by the Imperial Government of Ethiopia. It is strictly forbidden for any one to enter into such contracts or agreements.

PERMITS

Prospecting Permits

Prospecting is permitted only to individuals or companies having a permit, or permits, delivered by the Ethiopian Government. (Art. 3, decree of Apr. 8, 1928.) One person may apply for several permits at the same time; and if he wishes the country explored by several parties, or caravans, he must obtain a personal permit for the head of each group. Such a permit must be shown to local authorities upon demand. (Art. 6, decree of Apr. 8, 1929.)

A prospecting permit does not carry with it an exclusive right, as the Government reserves the right to grant permits to several different persons for the same region. (Art. 13, decree of Apr. 8, 1929.)

Application.- An application for a prospecting permit must be made in writing to the Department of Mines (at Addis Ababa), which will acknowledge receipt thereof and register it, with the date of its filing. Every application must contain:

1. The names, surnames, profession, nationality, and residence of the applicant and of the representative specifically designated; or if the applicant is a company, its name, its style of firm, its head office, and full information concerning its representatives at Addis Ababa.

2. Proofs of all declarations (extracts from the Register of Commerce, statutes, passports, et cetera).

3. Indication of the regions for which the permit is requested.

4. A statement of the minerals for which search is proposed.

The Government shall reply to the application within 60 days, either by refusing it (stating reasons for refusal not being obligatory) or by remitting a permit (with a copy of the decree of April 8, 1928) upon receipt of the required fee. (Art. 4 and 5, decree of Apr. 8, 1928.)

Area covered.- The mining decree sets no limit upon the size of the area that may be covered by a permit, but each permit must designate a definite area.

Duration.- A permit is granted for one year and is subject to several renewals for the same period. (Art. 7, decree of Apr. 8, 1928.)

Fee.- A fee of 30 thalers is due for each permit and for each renewal thereof, payable upon delivery. (Art. 8, decree of April 8, 1928.) At normal rates of exchange, the fee would be equal to $15 in United States currency.

Damages.- The owner of a prospecting permit must indemnify the proprietor of the land for all damages caused by his work. (Art. 10, Decree of April 8, 1928.)

Disposal of minerals.- Without the consent of the Government, the owner of a prospecting permit may not dispose of the minerals uncovered by his operations, although he is free to analyze and test them. Violations of this restriction entitles the Government to claim double the value of the minerals of which illegal use has been made. (Art. 11, decree of Apr. 8, 1928.)

Lands exempted.- Prospecting is forbidden on public ways and public places, on railway lines, within cemeteries, and within 250 meters of fortified places; prospecting is forbidden except with the consent of the proprietor of the surface within 25 meters of houses, churches, stations, and inclosed grounds. (Art. 9, decree of Apr. 8, 1928.)

Conditions imposed.- Every holder of a prospecting permit submits definitely to the following conditions (art. 14, decree of Apr. 8, 1928):

1. Not to disturb the work of exploitation on existing mining concessions.

2. Not to permit prospecting to become exploiting (mining).

3. Not to sell or cede, even gratuitously, either his permit or any of the rights and advantages accruing thereto.

4. To submit to administrative surveillance, especially with respect to the safety of the workmen, prevention of accidents, and like matters.

5. To bear full responsibility for all accidents that are the result of lack of foresight or surveillance and to make without delay a detailed report to the Department of Mines of each serious accident that occurs.

The holder of a permit is authorized to do only surface prospecting. Test pits with a maximum depth of 3 meters may be sunk on lands belonging to the Government, but not on private property except with the consent of the proprietor or holder thereof. (Art. 12, decree of Apr. 8, 1928.)

The holder of a permit shall make no claims against the Ethiopian Government for reimbursement of the cost of the permit or for any indemnity whatsoever in case the prospecting remains fruitless. (Art. 15, decree of Apr. 8, 1928.)

If the holder of a permit (or his employees or his workmen) shall discover archeological or paleontological objects, he shall notify the Department of Mines of the discovery and shall remit gratuitously to the Government the objects discovered. (Art. 16, decree of Apr. 8, 1928.)

Steps necessary to preserve rights.- In order to preserve his right to a mining privilege by reason of a discovery made under a prospecting permit, the discoverer must (art. 17, decree of Apr. 8, 1928) act as follows:

1. Indicate the place of discovery with a sign bearing his name, the date and the hour of its posting, and the nature of the exploitable mineral. (To insure secrecy, the sign may be placed in a bottle or a soldered box, which may be buried at the place of discovery.)

2. Address immediately to the Department of Mines an application for either a permit of search (giving the right to make deep excavations) or a permit of exploitation (mining).

3. Keep in good condition the identifying sign until the Government shall have passed upon the application, which must be accompanied by:

(1) A detailed description of the position of the deposit.

(2) A map or sketch of the region for which the permit is requested, as well as an accurate indication on the map or sketch of the perimeter desired, the coordinates defining its center, its extent, and the true north. (If the notification of discovery has been buried, the hiding place of the bottle or box must be indicated on the plan.)

(3) An indication of the nature of the mineral discovered or a sample thereof.

Form of permit.- A prospecting permit shall have the following form (art. 20, decree of Apr. 8, 1928):

No. of order _____.

The owner of the present permit, Mr. ____________________
(names, surname, and profession) ____________________,

or

The company ________________ (name and head office) _________, represented by Mr. ______________ (names, surname, and profession) ______________, nationality _____________, residing at __________, is authorized by the present permit to travel freely over the region of _______________ to make geological and mining studies or explorations according to the clauses and conditions of the decree regulating the prospecting of mines within the Empire of Ethiopia, promulgated on Miazia 1, 1920 (April 8, 1928).

Mr. _____________ is authorized to make explorations on the surface and down to a depth of 3 meters and to take samples or ores and, if he has made a discovery, to place a sign on the spot of the discovery and to station a guard.

This authorization is granted to Mr. ____________________ for a period of one year, beginning on _______________, and ending on ______.

Mr. _____________ declares that he understands the decree regulating the prospecting of mines, dated Miazia 1, 1930 (April 8, 1928), and engages himself to conform to it in all respects.

The chiefs of the districts of _________________ must give proper reception to Mr. _______________, and must not prevent his work, and must give him every facility to conduct his explorations to a successful end.

Addis Ababa, ___________________, A. D.

Mining Permits

The decree of April 8, 1928, which makes the foregoing provisions concerning prospecting, has little reference to mining, or exploiting, except to state that the Department of Mines has the right to determine the conditions under which applications for deep excavating and exploiting privileges (filed by prospecting permittees that are also discoverers) shall be granted.

The Department of Mines (which must keep a register of all applications, with the date and the hour of their filing) must give preference to the applicant first filing his application when applications are made for the same place and the same mineral. The Government must render its decision within three months of the presentation of an application. (Art. 18 and 19, decree of Apr. 8, 1928.)

The new mining laws and regulations of comprehensive character that are now under consideration by the Ethiopian Government may make specific rulings for mining concessions, but at present no definite limits are placed upon the size of a concession, its duration, its renewal, or its cancellation. A concession may be transferred only by special arrangement with, and by the consent of, the Government, which does not bind itself to approve the transfer.

As has been stated previously in this paper, conditions under which mining may be done are determined by the Emperor himself, each concession being treated individually.

A recent report, dated August 20, 1929, made by R. F. Chesbrough, an American trade commissioner at Cairo, Egypt, as discussed by Furness,[15] gives a few of the conditions attached to concessions granted comparatively recently (referred to in the introduction to this paper, under subsection entitled "Recently Reported Concessions"). In general, as Furness reports, concessions have provided for percentage royalties on production, to be paid to the Ethiopian Government. This participation may be in the form of a fixed stock interest or a percentage of the profits. Certain details are as follows:

> Gold and platinum concessions in the Province of Beni Shangul, granted to the Deutsches Studiensyndikat, of Stuttgart, Germany, are 5 by 5 kilometers square and cover a total area of 900 square miles. The concessions permit prospecting and exploiting of all mineral deposits in the allotted district. They were granted by the Emperor and ratified by the German minister at Addis Ababa, but it was necessary to obtain the approval of the governor of the Province to operate in the areas granted. (Through the decree of March 5, 1931, the approval of no one except the central Government is now required.)
>
> Gold and platinum concessions in the Provinces of Beni Shangul and Wollaga were allotted by the Emperor to an Italian citizen. The territory was granted in the form of seven circles, each circle having a radius of 5 kilometers. Ten per cent of the output was to be given to the Government.
>
> A third concession was granted by the governor of the Province of Sellale to a citizen of Ethiopia for deposits of saltpeter. This concession covers an area of 25,000 square kilometers. The terms of the grant allow the concessionaire to select seven different sections for exploitation, each of which is to have an area of 9 square miles. A royalty of 3 per cent on the nitrates extracted for the first two years is to be paid to the Government, the rate to increase to 6 per cent, beginning with the fifth year, and to

15 Furness, James W., Work cited.

apply thereafter until the expiration of the concession, which is to run for 30 years. At the option of the Government this concession may be renewed for a similar period. The concessionaire is given the right to make use of timber and water on the property without charge.

A contract, dated May 9, 1919, was made between the Ethiopian ruler and a French company, by which the company was to exploit (in the stead of the Government) for 75 years minerals found in Ethiopian territory, except in lands granted to other concessionaires. This contract, which represents one of the earlier attempts to develop a mining industry, contains the possibility of conflict with later contracts, as well as conflicting elements within itself. For example, the following provision is made: After the company has discovered the land for working and has advised the Ethiopian Government, no one shall have the right to make researches or excavations within 20 kilometers of the boundary indicated. However, should the company fail to begin the work of exploitation within four years of the discovery of any mineral, the Government may keep the mine for itself or grant it to another. The possibility of contracts with other concessionaires is further indicated by a clause providing that should the Government grant to another society more favorable conditions than those granted under the contract in question the French company shall benefit equally.

Among the various provisions of the contract are the following:

The French company was authorized to appeal for foreign, as well as Ethiopian, capital and to form companies with foreigners, as well as with Ethiopians, but not with foreign governments. If the Ethiopian Government chose to invest capital in the company, it might do so; and Ethiopians were to have the right to furnish one-half or one-quarter of the total capital.

The company was to pay a tariff of 5 per cent ad valorem on all the minerals (with the exception of gold and silver) extracted and worked mechanically or chemically. With respect to gold, silver, and precious stones, if the Government desired to take the crude materials, the concessionaire was to receive 12 or 20 per cent of the net benefit. The Government was to have the first right of purchase for all minerals.(gold and silver and precious stones at the current prices in Ethiopia and all other minerals at the latest European prices). The remaining minerals might be sold locally or abroad. Should the Government desire to take more petroleum than was its due, the company should not make exports abroad until the Government had satisfied its needs; nor should nationals be prevented from purchasing the petrol they desired.

The concessionaire was to be liable to an annual tax of 10,000 thalers per "gacha" of land retained for mining. Should the Government prefer payment in mining products, the company was to dispose of these products to the best possible advantage (as if selling for itself), deduct the expenses of transport and insurance, and transmit the balance to the Government. The company was to pay for five years an annual inspection fee of 15,000 francs and thereafter a minimum of 15,000 francs.

The company was to be granted freedom from duties on instruments and supplies necessary for its work; the use of water from nearby rivers and timber from unoccupied lands; the right to construct the necessary railroads; and all reasonable protection from the Government.

EXTRALATERAL RIGHTS

As neither law nor precedent exists governing extralateral rights, any claim therefor probably would be disputed, although a special provision might be arranged for in any particular concession.

RENT AND ROYALTIES

The Government requires the payment of rent and royalties; however, the amount is not a fixed one but rather a matter of private agreement between the Emperor of Ethiopia and the concessionaire.

I. C. 6537

October, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING, TREATMENT, METHODS, AND COSTS AT THE EAST TEXAS GRAVEL CO.'S DEPOSITS NEAR BOIS D'ARC, TEX.

BY

Walter W. Hyde

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING, TREATMENT METHODS, AND COSTS AT THE EAST TEXAS GRAVEL COMPANY'S DEPOSITS NEAR BOIS D'ARC, TEX.[1]

By Walter W. Hyde[2]

INTRODUCTION

This paper describing the mining and preparation of sand and gravel for a number of special markets by the East Texas Gravel Co., Dallas, Tex., is one of a series being prepared on the subject by the United States Bureau of Mines.

These articles are designed to disseminate technical information regarding the methods used. The cost tabulations represent operating expenditures only and not total production costs. It is recognized that publication of total production costs might in many instances cause embarrassment to individual producers as well as to the industry as a whole, but on the other hand, operating costs are essential to the technical discussion and study of the methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue.

Operators of sand and gravel deposits who have a surplus of sand and small pebbles which must be sent to waste will be particularly interested in the description of equipment and the method of removing these waste materials near the excavating machine in the pit, while mining.

The gravel deposits are located in the southwestern part of Kaufman County about 20 miles southeasterly from Dallas and approximately 5 miles south of the main line of the Texas and New Orleans Railroad (Southern Pacific Lines), at Bois D'Arc. From this point a standard-gage railroad track has been constructed, connecting the main line with the loading tracks at the deposits.

ACKNOWLEDGMENTS

In the preparation of this paper the author wishes to acknowledge the assistance of J. R. Thoenen, mining engineer of the United States Bureau of Mines, and the officials of the East Texas Gravel Co. The author also wishes to express his gratitude to G. A. Parkinson, of the University of Texas, for information regarding tests of the gravel and the geology of the deposits.

HISTORY

In early operations a small dragline machine was used to remove the overburden and then to excavate the sand and gravel and load it on standard-gage railroad cars. The material was used for railroad ballast, but the demand for other uses grew so rapidly it was soon necessary to increase production, and the present plant is the result of a gradual development brought about by the necessity of producing various special grades of sand and gravel.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6537."

2 - One of the consulting engineers, U. S. Bureau of Mines.

DESCRIPTION OF DEPOSITS

The sand and gravel deposits, which lie in the immediate vicinity of Dallas and in close proximity to the Trinity river and its tributaries, are quite similar in character and as a general rule are all more or less subjected to overflow of flood waters.

The deposits were derived from the disintegrated and broken fragments of the hard limestone and sandy formations of the Commanche and Carboniferous rock to the west. Their origin is proved conclusively by the character of the constituents and by the fossils found in the deposits.

Unquestionably these materials were worked downstream, largely by flood waters, to form these gravel deposits, and the materials while in transit were not only rounded by abrasion but were to a considerable extent segregated by size in the swift currents, so that from point to point it is noticeable that the pebbles and grains of sand differ in shape as well as in size. Often the dividing line between two or several adjoining deposits is remarkably well defined.

The Trinity river flowed over many different outcrops of rock, with the result that the amount and kind of materials contributed to the stream load have differed. This is exemplified by the difference between the upland gravel deposits and those found in the present flood plain of the river.

The upland gravel deposits are not as valuable as the flood-plain deposits as they carry great quantities of foreign materials, such as clay, mud, soft limestone, chalk pebbles, etc., well mixed with rather poor grades of sand and gravel. However, the raw material from these deposits makes excellent driveways or country roads where traffic is light, due to the natural binders contained in the gravel.

The flood-plain deposits where proved to contain material of sufficient quality and quantity to justify exploitation on a large scale are very valuable. They are usually free from foreign material such as soft limestone, chalk pebbles, mud, clay, and silt and therefore may be mined and made into a finished product with a minimum of expenditure.

Gravel deposits in this locality, and, for that matter, throughout northeastern Texas, are small, usually not exceeding 40 or 50 acres in area and averaging 20 to 25 acres. However, a number of these deposits are often found close together. The deposits are also comparatively shallow. The upland deposits have practically no overburden and are from 3 to 5 feet thick, whereas the flood-plain deposits are covered with 8 to 12 feet of soil and clay overgrown with trees and brush, and very from 8 to 20 feet in thickness. The deposits are usually underlaid with a hard blue shale or a stratum of solid limestone. However, in many cases the base of the gravel beds is a hard clay.

The beds themselves are composed of silica and limestone in the form of sand and pebbles both of which are uniform in texture throughout. The sand is angular and hard, and the pebbles are also hard, well formed, and of the right size for a number of uses.

PROSPECTING AND EXPLORATION

Locating gravel deposits on the upland, is a comparatively simple matter as the gravel sometimes outcrops or is usually detected in the soil within a few inches of the surface, or is exposed in gully washes along the banks of small streams that drain the high ground. Often the upland deposits are found accidentally in digging post holes.

Locating gravel deposits on the flood plains is more difficult, as the gravel is covered by 8 to 12 feet of clay and soil and prospecting is expensive as the clay is both hard and deep. Further, commercial deposits are seldom discovered, even though a prospector may spend months in sinking holes with a boring machine or digging prospect shafts.

When a deposit has been located, it is usually explored by means of test pits dug by hand to the top of the gravel. If the pit is dry, hand digging may be continued through the deposit. Where this method is used, one man to a hole is all that is necessary for the first 7 feet of depth. From this on, there are two men to the hole, which is usually 3 by 5 feet in cross section through the overburden.

One man on the surface lays a plank across the hole to stand on and uses a 1-inch rope (large enough to keep from burning the hands while gripping it), to hoist a steel bucket that holds about ½ cubic foot of gravel.

The man in the hole, digs and loads material into the bucket. When the surface of the gravel deposit has been reached, it is well to change from a 3 by 5 foot oblong hole to one that is round and about 3 feet in diameter. If the gravel is not too loose, the man in the hole will be safe in going down in the gravel to a depth of 10 to 12 feet, or to 20 to 25 feet below the surface. Should the gravel be extremely loose, a specially constructed steel curb, in sections, should be used for the protection of the man in the hole.

One gravel company in this vicinity has a boring machine, similar to a post-hole auger on a large scale, that is mounted on and operated by a Fordson tractor. This machine bores a hole about 16 inches in diameter through the clay overburden at the rate of 1 foot to 5 feet per minute, depending on the hardness of the clay; and in case the hole caves, the auger can be pulled by reversing the direction of rotation.

The machine drills in clay to a depth of about 18-feet without difficulty, but it does not work so well in the gravel because of the continual falling of pebbles, which causes many cave-ins. For this reason drilling is stopped immediately upon striking the gravel, and the hole is continued by using a small orange-peel bucket, operated by hand, inside a 14-inch steel casing. This is a good outfit for working in loose gravel above the water line. When the water is struck, faster progress may be made by using a flap-valve bailer (bailing bucket used in bailing oil wells, or other deep wells). The bailer used for sinking through gravel under water is operated inside of a 10-inch pipe. It is usually 8 inches in diameter and is fitted on one end with a 3-pronged, hard-tempered cutting edge. Should the bailer be operated by hand, it should not be more than 2 feet long; as many as 8 men, working 6 at a time, will be needed to make the best progress.

The bailer is suspended from a light derrick, mounted on a sled, and is worked up and down by a 1-inch rope running through a pulley at the top and one near the base. The derrick on the sled is moved by team, tractor, or by the crew of 8 men.

The bailer when used on a self-moving well-drilling machine of the cable type, has proved successful in testing gravel deposits which lie below the water. Two men can operate this equipment to good advantage, and they can usually drill from 70 to 80 feet in 10 hours.

The gravel deposits near Bois D'Arc, now being worked by the East Texas Gravel Co., are typical flood-plain deposits with all the peculiar characteristics so noticeable in the many other similar deposits lying close to the Trinity River which are being worked by other operators.

The East Texas Gravel Co.'s deposits cover several hundred acres and the overburden of clay ranges between 4 and 12 feet. The sand and gravel have a depth of from 10 to 15 feet. The pit-run material averages about 45 per cent sand and 55 per cent gravel, and is fairly clean. The deposit lies below the normal water level which comes up to the base of the clay overburden.

The materials, both sand and gravel, in these deposits, are of exceptional quality; being free from mud, clay, soft limestone, chalk pebbles, etc., and particularly free from a medium-soft white limestone pebble which is mixed in small amounts with the sand and gravel from pits some 50 miles further up the Trinity River. The sand is angular, coarse, and hard, and is prepared by washing and sizing so that it meets practically all specifications, especially for concrete construction of different kinds.

The gravel is hard and tough, and particularly well adapted, because of size and quality, to use in concrete structures such as building columns, side walls, floor slabs, and all kinds of highway-drainage structures. It also fully meets specifications required by the Texas Highway Department and Federal engineers for the construction of concrete slabs and hard-surface tops of permanent highways.

The gravel and sand are thoroughly washed and rinsed, and sized to meet specifications for many different projects.

Only a small percentage of the gravel exceeds 3 inches in size, and for this reason no crushing is necessary. All of the oversize is sold to the railroads, or to contractors who do their own crushing on special jobs.

A screen analysis on an average sample of washed sand for concrete is as follows:

	Screen	Per cent
	Inch	
On	½	00.0
On	¼	18.0
Through	¼	82.0
Total		100.0
	Mesh	
On	10	33.6
On	20	54.2
On	30	76.6
On	40	89.2
On	50	95.6
On	60	97.2
On	80	97.9
On	100	98.2
On	200	98.3
Through	200	1.7
Total		100.0

An analysis of an average sample of washed gravel for concrete is as follows:

	Screen	Per cent
	Inch	
On	2	00.0
On	1½	1.5
On	1¼	7.1
On	1	19.1
On	¾	45.3
On	½	72.7
On	¼	94.6
Through	¼	5.4
Total		100.0
	Mesh	
On	10	99.5
On	20	.5
Total		100.0

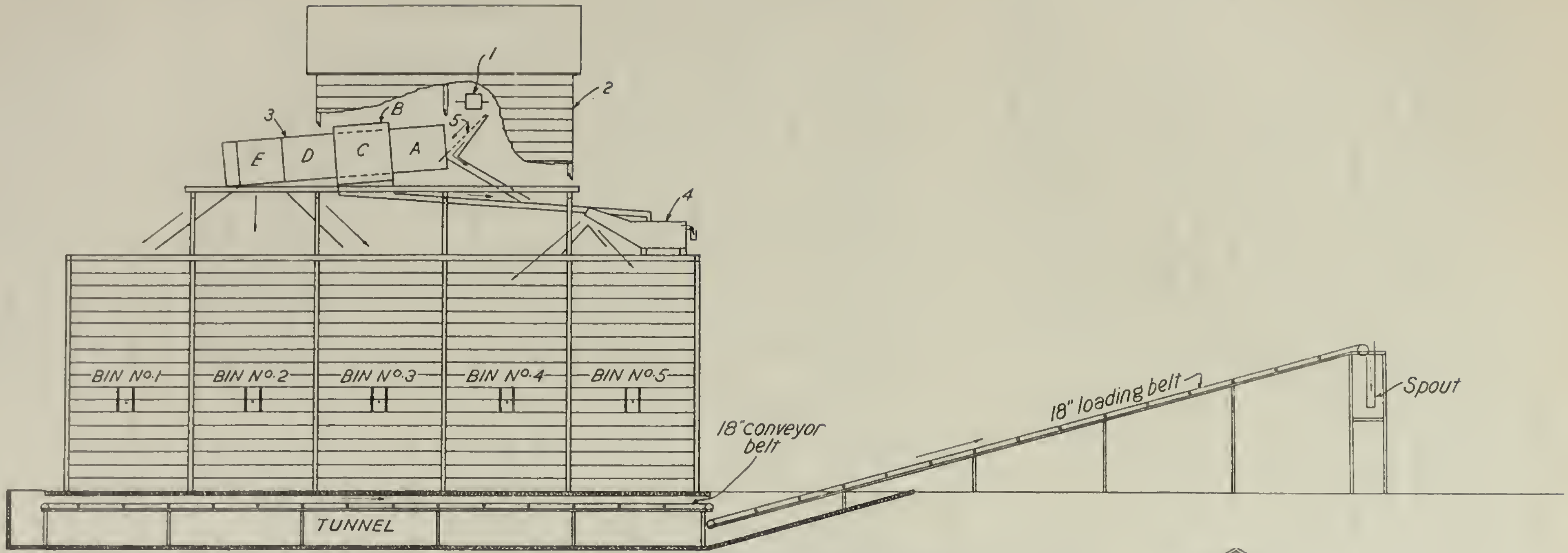

Figure 1.- Side elevation of bins and loading conveyor belt with longitudinal section of tunnel

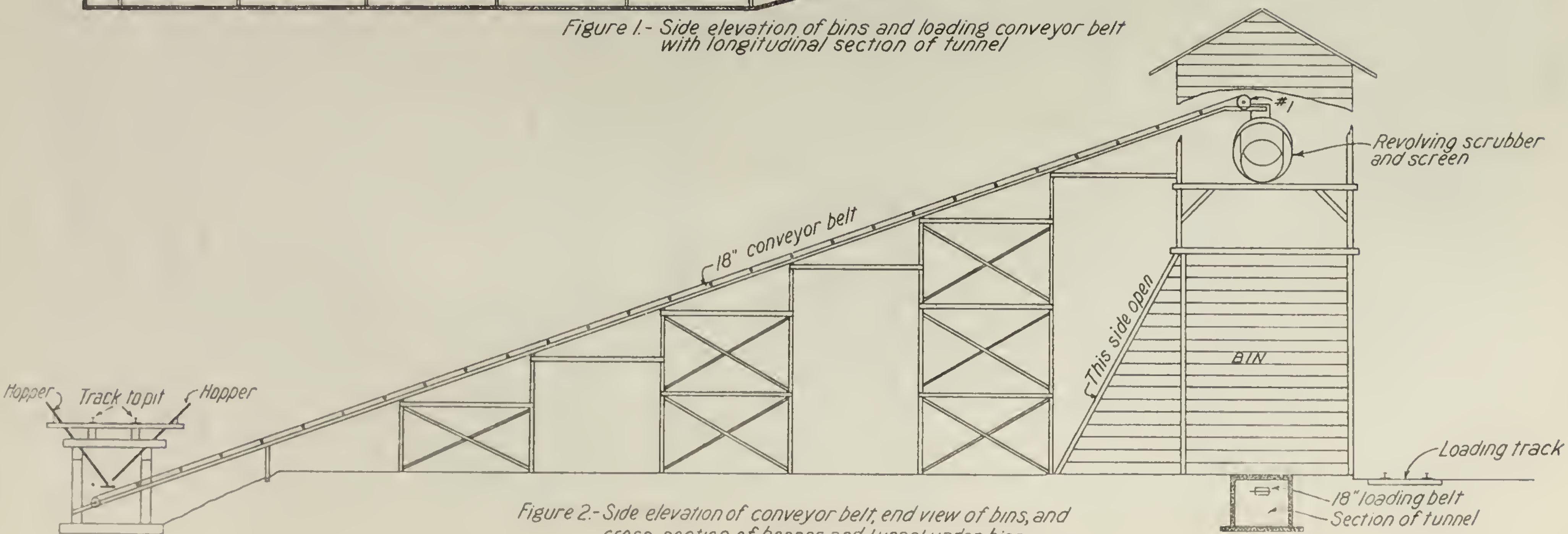

Figure 2.- Side elevation of conveyor belt, end view of bins, and cross section of hopper and tunnel under bins

STRIPPING AND MINING

Removal of clay overburden and excavation of raw pit gravel is done by a Monighan walking dragline with an 85-foot boom and a 3-cubic yard bucket. This machine is powered by a 175-hp. Fairbanks-Morse Diesel engine belted directly to a shaft with clutches connecting with two drums on the hoist used for dragging and hoisting the bucket. There is also a belt to an electric generator which supplies current for a 50-hp. motor directly connected by pinions and gears to a circular rack for swinging the machine.

The dragline is put to work at any desired location in the pit, and first removes the overburden of clay which it casts to one side, usually at about 90° from the direction of the loading truck. After the overburden has been removed from about 5,000 square feet, the dragline proceeds to remove the pit gravel and places it in standard-gage, bottom-dump, 50-ton steel cars, for transport to the Texas and New Orleans Railroad Co.'s track to Bois D'Arc. From there it is transported to designated points along the main line and used for ballast.

The ballast cars are often utilized for transporting the pit gravel as loaded by dragline to the hopper at the washing plant. The ballast cars are spotted, one at a time, over the hopper and dumped.

SCREENING PLANT

The raw material is fed from the hopper by a plate feeder on to an 18-inch belt conveyor that carries the material to the top of the washing plant over a head pulley shown in Figures 1 and 2. Figure 1 shows a side elevation of the washing plant and a longitudinal section of the tunnel under the bins. Figure 2 shows a side elevation of the conveyor belt from the hopper, end view of bins, and cross section of hopper and tunnel under bins. The top portion of the house which covers the belts and electric motors that drive the 18-inch conveyor belt, revolving scrubber, and screen is shown at 2, Figure 1.

From the head pulley (1) at the top of the plant the pit gravel is sluiced over an inclined screen (5) of 1/8-inch steel wire having 3/8-inch square openings.

The undersize from this goes directly to the two sand classifiers (4) thus relieving the revolving screen of this burden and thereby speeding up production.

The oversize from the inclined screen is flushed into the revolving scrubber and screen (3) which is 18 feet long and 4 feet in diameter, and is surrounded midway by a wire jacket, B, that is 6 feet in diameter and 9 feet long. The jacket is made of 1/8-inch steel wire with 3/8-inch square openings for the first 5 feet of its length and 1/2-inch square openings for the remaining length of 4 feet. The scrubber portion of the revolving screen A, is on the receiving end and is 5 feet long. It is followed by sections C, D, and E, which are 6, 4, and 3 feet long and have 5/8, 1-3/8, and 3-inch round perforations, respectively.

The sand and water passing through the screen jacket are sluiced to two sand classifiers (4), placed side by side. The classifiers wash, size, dewater, and deposit the sand in bins 4 and 5. The sand classifiers are of the box type, with long sloping necks in which sand drags are operated. Each sand drag consists of two parallel chains mounted on cast-steel sprocket wheels and connected by steel plates 3/8 inch thick, 5 inches wide, and 30 inches long.

The sand drags, driven by gear and pinion, agitate the sand and keep dirt, silt, etc., in suspension to float off with the water, while the sand is dewatered and dragged out through the neck of each classifier.

Returning to the revolving scrubber and screen, we find that the inclined screen ahead of it and the jacket have removed most of the sand and water, leaving only the pebbles to be screened into three sizes and chuted to their respective bins.

Referring to Figure 1, bin 3 is for pebbles that pass through a 5/8-inch ring and are retained on 3/8-inch square mesh opening. Bin 2 is for pebbles through 1-3/8 inches and over ½ inch. Bin 1 is for pebbles through 3 inches and over 1-3/8 inches. The different sizes may be loaded out individually into cars through the gates on the side of each bin, or may be remixed in any desired proportion in a trough immediately under the revolving screen and from there spouted directly to the cars on the loading track for shipment.

In many instances where it is necessary to be more particular about the proportions of the several sizes, mixing is accomplished by use of the loading belt as follows:

Underneath the bins as shown in Figures 1 and 2, there is a tunnel having a 7 by 7 foot inside cross section and running parallel with the longitudinal axis of the plant. At intervals along the roof of tunnel, which forms a portion of the bottom of the bins, there are gates spaced from 10 to 12 feet apart along the center line. An 18-inch conveyor belt traverses the entire length of the tunnel, and the flow from the bins onto the conveyor belt is regulated by adjusting the bin gates to produce any desired specification. Small hoppers straddling the conveyor belt are placed beneath the various bin gates to guide the discharge.

After passing out of the tunnel the gravel is discharged onto another conveyor belt which rises to a point high enough to permit spouting the material over an inclined screen where it is sprayed with water with sufficient force to thoroughly rinse off the gravel before it is finally placed in the car for shipment.

Using the small dragline machine at the washing plant to stack pea gravel and sand in stock piles, as well as to reclaim it and load it out quickly, has been found to be a most advantageous method. When the sand and pea gravel have been stocked in piles near the washing plant to the capacity limit of open storage, the handling and rehandling of these materials eventually became a problem, for in addition to being hauled to the plant and carried through the regular process of being washed it was necessary to load them from bins to cars or from storage to cars and finally to dispose of them by dumping as waste somewhere along the line.

At first this was not a bad arrangement, as these materials when mixed make an excellent track ballast, and were utilized for this purpose. However, when all the track had received an adequate amount of this ballast, the continued dumping of these materials became not only a problem of finding a place to dump them, but also an expensive operation.

The disposal of the pea gravel and excess sand eventually became an acute problem and a method was worked out to dispose of all undesirable materials at the pit by removing them from the desirable materials and discharging them as waste into the pond formed by previous workings.

This method utilizes a portable loading and washing platform loaded by the Monighan walking dragline for eliminating the undesirable materials, such as large stones and big chunks of clay or mud, etc., as well as pea gravel and sand that is overabundant in the pit gravel.

Large stones and big chunks of mud and clay are removed from the platform by hand, and the sand and pea gravel are washed away by water that sluices them through an inclined screen and thence by flume back to the excavated areas of the pit. The oversize on the inclined screen rolls into standard-gage steel ballast cars for transport to the washing plant. This method of treatment eliminates the haul on waste material, relieves congestion at the plant and speeds up production of those products for which there was a ready market.

Figure 3 shows how the portable platform is constructed.

It may be seen that the standard-gage ballast cars pass under the platform, and that the floor of the platform slopes two ways, toward the center and also toward an inclined screen at one end. The raw pit gravel is placed directly by dragline onto the platform floor, which is covered with 10-gage steel plates laid in "rainproof" fashion.

Across the inclined screen and some 18 inches above it, several 3-inch pipes are placed, each perforated on one side with $\frac{1}{4}$-inch holes about 1-inch apart, through which water under pressure is sprayed on to the screen.

When the raw pit gravel is placed on the platform the water which is played on it through a small hydraulic giant floods the gravel and sand down grade to the inclined screen which has a pitch of about 45°. This pitch is sufficient to make the oversize roll off into the car while the sand and pea gravel are washed through the screen by the force of the water sprays to be flushed through a flume, made up of 25-foot portable sections, to the worked-out portion of the gravel pit where it may be picked up again on the next pass with the dragline or left as waste along with the overburden.

To move the portable platform, four timbers each 8 by 8 inches by 12 feet, a few blocks each 8 by 8 inches by 2 feet, and two good track jacks, are required to raise and block it upon one of the ballast cars. A special pocket was built into the portable platform to hold the timbers, blocks, and jacks so that they are always available when needed.

One of the ballast cars is used for a moving van, so to speak, by spotting it on the loading track directly under the platform. The four timbers are then placed across the car, two at each end of the platform immediately under the horizontal cross timbers of the platform that are about 18 inches above the top of ballast car. Both jacks are set, and one end of the platform is raised and blocked on top of the car, leaving its legs at that end some 8 inches or so above the foundation blocks. In like manner the other end of the platform is raised and blocked and the whole structure, then clear of all foundation blocks and resting on the ballast car, may be moved by the locomotive to a new location for further use or it may be taken to the end of the loading track to remain completely out of the way until it is again needed.

Water for use on the portable platform is plentiful, there being 10 or more feet of it standing in the pit at all times. A 6-inch centrifugal pump powered by a 30-hp. gasoline engine is used to pump the water through a 6-inch pipe to the top of the platform. The centrifugal pump and gasoline engine remain more or less stationary, but the 6-inch pipe laid parallel to and about 10 feet away from the track, is shifted each time the loading track is moved over for the new pass of stripping and loading.

SUMMARY OF COSTS

Period: July, 1929 to July, 1930.
Total material loaded during period:
Overburden, 110,000 cubic yards moved by dragline into pond formed by previous workings.
Sand and gravel, 123,500 tons.

Operating costs per ton of sand and gravel mined

	Labor	Supervision	Power	Fuel	Other supplies	Total
Operating costs:						
Stripping	$0.025	$0.002	–	$0.010	$0.022	$0.059
Mining and loading	.049	.005	$0.002	.002	.051	.109
Transportation	.032	.001	–	.014	.002	.049
Washing, screening and loading	.080	.010	.002	–	.003	.095
Storage	.005	–	–	.001	.005	.011
Repairs and maintenance	.020	.001	–	–	.110	.131
Miscellaneous	.006	–	–	–	.005	.011
Total	0.217	0.019	0.004	0.027	0.198	0.465

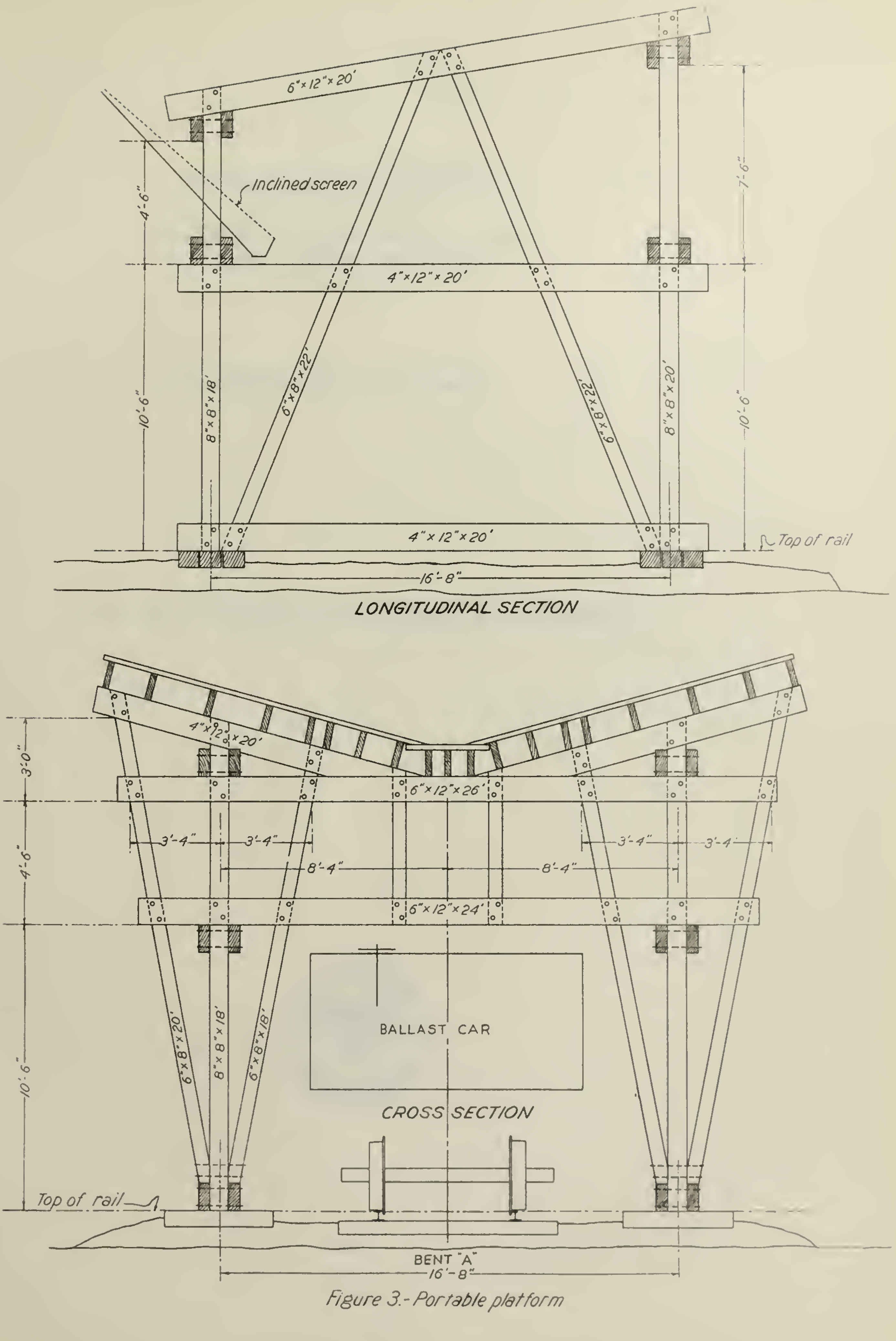

Figure 3.- Portable platform

I. C. 6538

November, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

LIST OF PERMISSIBLE MINE EQUIPMENT

BY

L. C. Ilsley

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

LIST OF PERMISSIBLE MINE EQUIPMENT[1]

By L. C. Ilsley[2]

A complete list of permissible mine equipment, tested and approved prior to January 1, 1931, was published in Bureau of Mines Information Circular 6443.[3] The present list[4] includes practically all the electric air compressors, coal drills, mining machines, loading machines, conveyors, mine pumps, room hoists, mine telephones, rock-dusting machines, switches, electric cap lamps, flame safety lamps, electric hand and trip lamps, electric flood lamps, methane detectors, blasting units, storage battery locomotives, tandem locomotives, power trucks, and concrete mixers tested and approved by the bureau up to July 1, 1931.

The system under which these devices were tested permits the manufacturer, after his equipment has passed certain tests prescribed by the United States Bureau of Mines, to mark his equipment with a seal showing that it has been approved by the bureau. These tests are designed to insure that equipments have the minimum requirements for safety in use. The only object of the bureau in making such tests and publishing lists of permissible equipment is to help lessen the hazards of mining.

PERMISSIBLE MINING MACHINES, COAL DRILLS, ETC.

Approved Under Schedules 2, 2A, 2B, and 2C

Air Compressors

1. Type WK-26 compressor; 30-hp. motor, 250-500 volts, D. C. Approvals 117 and 117A, issued to Sullivan Machinery Co., March 12, 1925.

2. Type WK-39 self-propelled compressor; 30-hp. motor, 250-500 volts, D. C. Approvals 120 and 120A, issued to Sullivan Machinery Co., July 28, 1925.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6538."
This circular includes an appendix in which are listed such trailing cables as have met the bureau's requirements for "Specially Recommended Cables."
2 Electrical engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.
3 Ilsley, L. C., List of Permissible Mine Equipment: Inf. Circ. 6443, Bureau of Mines, May, 1931, p. 14.
4 A few equipments which may now be considered obsolete and therefore not obtainable are not included in this list.

3. Type CP-26G, CP-26D, and CP-36H compressors; 25-hp. motor, 250-500 volts, D. C. Approvals 128 and 128A, issued to General Electric Co., March 21, 1927, and July 16, 1926, respectively.

4. Type 20 compressor; 15-hp. motor, 250-500 volts, D. C. Approvals 159 and 159A, issued to Ingersoll-Rand Co., Sept. 4, 1928.

5. Type WK-22 compressor; 30-hp. motor, 250-500 volts, D. C. Approvals 160 and 160A, issued to Sullivan Machinery Co., Sept. 10, 1928.

6. Type WK-22 compressor; 20-hp. motor, 230 volts, D. C. Approval 189 issued to the Sullivan Machinery Co., April 22, 1930.

Loading Machines and Conveyors

1. Type 43-A shortwaloader; 50-hp. motor, 250-500 volts D. C. Approvals 122 and 122A issued to The Jeffrey Manufacturing Co., Jan. 8, 1926.

2. Type 44-B conveyor-loader; 50-hp. motor, 250-500 volts, D. C. Approvals 123 and 123A, issued to the Jeffrey Manufacturing Co., Jan. 15, 1926.

3. Belt-type conveyor; 5-hp. motor, 250 volts, D. C. Approval 126, issued to Bird Coal Co., June 25, 1926.

4. Shovel-type loading machine; 30-hp. motor, 250-500 volts, D. C. Approvals 127 and 127A, issued to Myers-Whaley Co., July 16, 1926, and Sept. 23, 1927, respectively.

5. Chain-type conveyor; 5-hp. motor, 250 volts, D. C. Approval 129, issued to the Bird Coal Co., July 21, 1926.

6. Type 5-BU loading machine; 25-hp. motor, 230-500 volts, D. C. Approvals 132 and 132A, issued to Joy Manufacturing Co., December 29, 1926, and March 22, 1927, respectively.

7. Type 49-A chain-type conveyor; 3-hp. motor, 250-500 volts, D. C. Approvals 133 and 133A, issued to The Jeffrey Manufacturing Co., Feb. 10, 1927.

8. Conveyor-type loader; 30-hp. motor, 250-500 volts, D. C. Approvals 135 and 135A, issued to the Sullivan Machinery Co., May 11, 1927.

9. Type 136-EC entry loader; 35-hp. motor, 210-500 volts, D. C. Approvals 138 and 138A, issued to the Goodman Manufacturing Co., Aug. 5, 1927.

10. Belt-type conveyor; 1-hp. motor, 220 volts, D. C. Approval 139, issued to the Lorain Steel Co., Aug. 19, 1927.

11. Type A.F.-10 by 8 shaker conveyor; 15-hp. motor, 250-500 volts, D. C. Approvals 149 and 149A, issued to C. H. McCullough Engineering Co., March 29, 1928.

12. Type 48-E power shovel; 15-hp. motor, 210 volts, D. C. Approval 150, issued to Goodman Manufacturing Co., May 11, 1928.

13. Chain-type conveyor; 5-hp. motor, 250 volts, D. C. Approval 151, issued to South Fork Foundry & Machine Co., May 19, 1928.

14. Conveyor-type face loader; 25-hp. motor, 250 volts, D. C. Approval 155, issued to Bethlehem Steel Co., August 2, 1928.

15. Shaker-type car filler; 22-hp. motor, 250 volts, D. C. Approval 156, issued to Bethlehem Steel Co., August 2, 1928.

16. Pit-car loader; (turntable type) 1-hp. motor, 250-500 volts, D. C. Approvals 166 and 166A, issued to Duncan Foundry and Machine Works, (Inc.), March 13, and July 11, 1929, respectively.

17. Brownie pit car loader; 1 1/2-hp. motor, 250-500 volts D. C. Approvals 167 and 167A, issued to the Brown-Fayro Co., March 27, 1929.

18. Pit car loader; 1-hp. motor, 230-500 volts D. C. Approvals 168 and 168A issued to the Northern Conveyor and Manufacturing Co., April 5, and Sept. 27, 1929, respectively.

19. Conveyor Sales Co. Shaker conveyor; 10-hp. motor, 250 volts, D. C. Approval 171 issued to the Goodman Manufacturing Co., April 17, 1929.

20. Pit car loader; 1 1/2-hp. motor, 250-500 volts, D. C. Approvals 173 and 173A issued to the Fairfield Engineering Co., April 30, 1929.

21. Pit car loader; 1-hp. motor, 230 volts, D. C. Approval 174 issued to Bertrand P. Tracy Co., May 22, 1929.

22. Pit car loader; 1-hp. motor, 250-500 volts, D. C. Approvals 175 and 175A issued to the Chicago Automatic Conveyor Co., July 26 and June 24, 1929, respectively.

23. Pit car loader; 2-hp. motor, 230 volts, D. C. Approval 178 issued to the Northern Conveyor and Manufacturing Co., Oct. 5, 1929.

24. Pit car loader; 1 1/2-hp. motor, 250-500 volts, D. C. Approvals 179 and 179A, issued to the Mancha Storage Battery Locomotive Co., Nov. 26 and Oct. 19, 1929, respectively.

25. Type L-2 loading machine; 13 motors aggregating 117-hp., 250 volts, D. C. Approval 182, issued to the Oldroyd Machine Co., December 9, 1929.

26. Type 148-E power shovel; 22-hp. motor, 220 volts, D. C. Approval 186 issued to Goodman Manufacturing Co., March 15, 1930.

27. Pit car loader; 1-hp. motor, 250-500 volts, D. C. Approvals 187 and 187A issued to the Bertrand P. Tracy Co., March 19, 1930, and October 17, 1930, respectively.

28. Type 44-C loading machine; two 7 1/2-hp. motors, 250-500 volts, D. C. Approvals 194 and 194A, issued to The Jeffrey Manufacturing Co., June 6, 1930.

29. Type 636 AK-3 entry loader; 35-hp. motor, 220-440 volts, A. C. Approvals 196 and 196A, issued to the Goodman Manufacturing Co., September 29, 1930, and July 26, 1930, respectively.

30. Pit car loader (Type 38D); 3-hp. motor, 220-440 volts, A. C. Approvals 200 and 200A issued to The Jeffrey Manufacturing Co., August 28, 1930.

31. Type G and Mat Type face conveyor; 2-hp. motor, 230 volts, D. C. Approval No. 203, issued to Gellatly & Co., Inc., September 23, 1930.

32. Type A conveyor; 5-hp. motor, 230 volts, D. C. Approval No. 205, issued to Gellatly & Co., Inc., October 30, 1930.

33. Face conveyor; 3-hp. motor, 230 volts, D.C. Approval No. 209 issued to the Fairmont Mining Machinery Co., December 2, 1930.

34. Type A conveyor; 5-hp. motor, 230 volts, D. C. Approval No. 212, issued to Gellatly & Co., Inc., December 26, 1930.

35. Type 44-C loading machine; two 7-1/2-hp. motors, 220-440 volts, A. C. Approvals 217 and 217A, issued to The Jeffrey Manufacturing Co., February 27, 1931.

36. Gellatly & Co. conveyor; 2-hp. motor, 500 volts, D. C. Approval 219A issued to Gellatly & Co., Inc., March 11, 1931.

37. Type VT(9 x 9-1/2) shaker conveyor; 15-hp. motor, 440 volts, A. C. Approval 221A issued to Mavor and Coulson (Ltd.), April 22, 1931.

38. Type 148-K3 power shovel; 35-hp. motor, 220 volts, A. C. Approval 222 issued to Goodman Manufacturing Co., May 8, 1931.

39. Type VT(12 x 9) belt conveyor; 15-hp. motor, 440 volts, A. C. Approval 224A issued to Mavor and Coulson (Ltd.), June 19, 1931.

Coal Drills

1. Type 2-BF drill; 1-hp. motor, 80,110, and 250 volts, D. C. Approvals 109 and 109A, issued to Chicago Pneumatic Tool Co., Sept. 19, 1922.

2. Type CD drill; 3/4-hp. motor, 110-230 volts, D. C. Approvals 110 and 110A, issued to Martin-Hardsocg Co., Sept. 16, 1922.

3. Type A-5 drill; 3-hp. motor, 110-250 volts, D. C. Approvals 119 and 119A, issued to The Jeffrey Manufacturing Co., April 15, 1925.

4. Type 56-A drilling machine; 2-type A-5 Jeffrey drills on truck, propelled by 13-hp. motor, 250-500 volts, D. C. Approvals 147 and 147A, issued to The Jeffrey Manufacturing Co., February 8, 1929.

5. Model A drill; 1 3/4 hp. motor, 250 volts, D. C. Approval 154 issued to Ohio Brass Co., Aug. 1, 1928.

6. Class CD-4 and CD-5 self-propelled coal-drilling machine; 5-hp. motor, 250-500 volts, D. C. Approvals 176 and 176A issued to Sullivan Machinery Co., September 9, 1929.

7. Class CD-4 and CD-5 self-propelled coal-drilling machine; 5-hp. motor, 250-500 volts, D. C. Approvals 177 and 177A issued to Sullivan Machinery Co., September 13, 1929.

8. Type AE-430 Siemens-Schuckert drill; 1/2-hp. motor, 250 volts, D. C. Approval 184 issued to the Colonial Supply Co., February 7, 1930.

Mining Machines

1. Type CE-7 shortwall mining machine; 30-hp. motor, 250-500 volts, D. C. Approvals 100 and 100A, issued to Sullivan Machinery Co., Sept. 30, and Oct. 20, 1914, respectively.

2. Types 12-CC and 12-EC shortwall mining machines, 35-hp. motor, 210-500 volts D. C. Approvals 101 and 101A, issued to Goodman Manufacturing Co., May 20, 1916.

3. Type 35-B shortwall mining machine, 35-hp. motor, 250-500 volts, D. C. Approvals 103 and 103A, issued to The Jeffrey Manufacturing Co., Nov. 2, 1917.

4. Type CE-7 shortwall mining machine, 30-hp. motor, 220-440 volts, A. C. Approvals 104 and 104A, issued to Sullivan Machinery Co., January 16, 1919.

5. Types 12-CJ and 12-EJ shortwall mining machines; 50-hp. motor, 210-500 volts, D. C. Approvals 105 and 105A, issued to Goodman Manufacturing Co., June 21, 1920.

6. Types 112-CC and 112-EC shortwall mining machines; 50-hp. motor, 210-500 volts, D. C. Approvals 106 and 106A, issued to Goodman Manufacturing Co., Feb. 9, 1922.

7. Types 12-CC and 12-EC shortwall mining machines; 35-hp. motor, 210-500 volts, D. C. Approvals 107 and 107A, issued to Goodman Manufacturing Co., Feb. 9, 1922.

8. Types 112-CJ and 112-EJ shortwall mining machines; 35-hp. motor, 210-500 volts, D. C. Approvals 108 and 108A, issued to Goodman Manufacturing Co., Feb. 9, 1922.

9. Type 35-BB shortwall mining machine; 50-hp. motor, 250-500 volts, D. C. Approvals 111 and 111A, issued to Jeffrey Manufacturing Co., Oct. 16, 1922.

10. Types 29-C, 29-D, and 29-E arcwall mining machine; 50-hp. motor, 250-500 volts, D. C. Approvals 112 and 112A, issued to Jeffrey Manufacturing Co., March 13, 1924.

11. Types 212-EJ and 212-CJ shortwall mining machines; 50-hp. motor, 210-500 volts, D. C. Approvals 113 and 113A, issued to Goodman Manufacturing Co., Nov. 4, 1924.

12. Types 112-CK3 and 112-EK3 mining machines; 35-hp. motor, 220-440 volts, A. C. Approvals 114 and 114A, issued to Goodman Manufacturing Co., Feb. 7, 1925.

13. Types 112-CL3 and 112-EL3 shortwall mining machines; 50-hp. motor, 220-440 volts, A. C. Approvals 115 and 115A, issued to Goodman Manufacturing Co., Feb. 7, 1925.

14. Type 124-EJ slabbing machine; 50-hp. motor, 210-500 volts, D. C. Approvals 118 and 118A, issued to Goodman Manufacturing Co., March 12, 1925.

15. Type 30-A shearing-drilling machine; 50-hp. motor, 250-500 volts, D. C. Approvals 125 and 125A, issued to The Jeffrey Manufacturing Co., April 26, 1926.

16. Type CLU cutting-shearing machine; 50-hp. motor, 250-500 volts, D. C. Approvals 134 and 134A, issued to the Sullivan Machinery Co., March 18, 1927.

17. Type CLE longwall mining machine; 30-hp. motor, 250-500 volts, D. C. Approvals 136 and 136A, issued to the Sullivan Machinery Co., May 28, 1927.

18. Type S-3 shearing machine; 25-hp. motor, 500 volts, D. C. Approval 148A, issued to Joy Manufacturing Co., Feb. 8, 1928.

19. Type B-2 mining machine; 50-hp. motor, 250-500 volts, D. C. Approvals 152 and 152A, issued to Oldroyd Machine Co., June 15, 1928.

20. Type 35-BB shortwall mining machine; 50-hp. motor, 220-440 volts, A. C. Approvals 153 and 153A, issued to The Jeffrey Mfg. Co., Aug. 1, 1928.

21. Types 224-EJ, 224-BJ, and 424-BJ slabbing machine; 50-hp. motor, 250-500 volts, D. C. Approvals 172 and 172A issued to the Goodman Manufacturing Co., April 29, 1929.

22. Type CLE-2 longwall mining machine; 30-hp. motor, 220-440 volts, A. C. Approval 181 and 181A issued to the Sullivan Machinery Co., December 2, 1929 and January 6, 1930, respectively.

23. Type 24-B longwall mining machine; 35-hp. motor, 220-440 volts, A. C. Approvals 185 and 185A, issued to The Jeffrey Manufacturing Co., February 24, 1930.

24. Type CLU cutting-shearing machine; 50-hp. motor, 220-440 volts, A. C. Approvals 188 and 188A, issued to the Sullivan Machinery Co., April 15, 1930.

25. Type CR-3 shortwall mining machine; 30-hp. motor, 220-440 volts, A. C. Approvals 191 and 191A issued to the Sullivan Machinery Co., May 21, 1930.

26. Type CR-3 shortwall mining machine; 30-hp. motor, 250-500 volts, D. C. Approvals 192 and 192A, issued to the Sullivan Machinery Co., June 3, 1930.

27. Type 35L shortwall mining machine; 50-hp. motor, 250-500 volts, D. C. Approvals 193 and 193A, issued to The Jeffrey Manufacturing Co., June 3, 1930.

28. Types 12EL-3 and 12CL-3 shortwall mining machines; 50-hp. motor, 220-440 volts, A. C. Approvals 197 and 197A, issued to the Goodman Manufacturing Co., July 31, 1930.

29. Types 12-EK3 and 12-CK3 shortwall mining machines; 35-hp. motor, 220-440 volts, A. C. Approvals 198 and 198A, issued to Goodman Manufacturing Co., August 1, 1930.

30. Type 35-L low vein cutting machine; 50-hp. motor, 220-440 volts, A. C. Approvals 201 and 201A, issued to The Jeffrey Manufacturing Co., September 8, 1930.

31. Types CS-4 Swivel-Shearer cutting machine; 30-hp. motor, 250-500 volts D. C. Approvals 202 and 202A, issued to Sullivan Machinery Co., September 17, 1930.

32. Type 312-E.J. shortwall mining machine; 50-hp. motor, 210-500 volts, D. C. Approvals 204 and 204A issued to the Goodman Manufacturing Co., October 13, 1930 and December 13, 1930, respectively.

33. Types 124-CJ and 324-CJ slabbing machines; 50-hp. motor, 210-500 volts, D. C. Approvals 207 and 207A, issued to the Goodman Manufacturing Co., November 14, 1930.

34. Type 524-E.J. slabbing machine; 50-hp. motor, 210-500 volts, D. C. Approvals 216 and 216A issued to the Goodman Manufacturing Co., February 12, 1931.

35. Type L-8CL3 longwall mining machine; 50-hp. motor, 220-440 volts, A. C. Approvals 218 and 218A issued to the Goodman Manufacturing Co., March 10, 1931.

36. "Samson" Type HT(12 x 11-1/2) longwall mining machine, 35-hp. motor, 440 volts, A. C. Approval 220A issued to Mavor and Coulson (Ltd.), April 21, 1931.

37. Type 312-EL3 shortwall mining machine; 50-hp. motor, 220-500 volts, D. C. Approvals 223 and 223A, issued to the Goodman Manufacturing Co., May 13, 1931.

Room Hoists

1. Oaks safety room hoist; 5-hp. motor, 250-500 volts, D. C. Approvals 116 and 116A, issued to South Fork Foundry and Machine Co., Feb. 13, 1925.

2. Brownie room hoist; 5-hp. motor, 250-500 volts, D. C. Approvals 162 and 162A, issued to Brown-Fayro Co., Nov. 30, 1928.

3. Type HDE-4 room hoist; 10-hp. motor, 250-500 volts, D. C. Approvals 164 and 164A, issued to Sullivan Machinery Co., Jan. 4, 1929.

4. Type HE-5 room hoist; 10-hp. motor, 250-500 volts, D. C. Approvals 165 and 165A, issued to Sullivan Machinery Co., Jan. 4, 1929.

5. Brownie model HG room hoist; 5-hp. motor, 250 volts, D. C. Approval 169, issued to the Brown-Fayro Co., April 5, 1929.

6. Room hoist; 5-hp. motor, 230 volts, D. C. Approval 190 issued to the Flood City Brass and Electric Co., April 29, 1930.

Mine Pumps

1. Fairmont No. 1 mine pump; 5-hp. motor, 115, 230, and 500 volts, D. C. Approvals 140 and 140A, issued to Westinghouse Electric & Manufacturing Co., Nov. 1, 1927.

2. Deming "Oil Rite" mine pump; 5-hp. motor, 115, 230, and 500 volts, D. C. Approvals 141 and 141A, issued to Westinghouse Electric & Manufacturing Co., Nov. 1, 1927.

3. Scranton 5 by 6 mine pump; 5-hp. motor, 115, 230, and 500 volts, D. C. Approvals 143 and 143A, issued to Westinghouse Electric & Manufacturing Co., Nov. 1, 1927.

4. Weinman "Self-Oiler" mine pump; 5-hp. motor, 115, 230, and 500 volts, D. C. Approvals 144 and 144A, issued to Westinghouse Electric & Manufacturing Co., Nov. 1, 1927.

5. Austin 5 by 6 mine pump; 5-hp. motor, 115, 230, and 500 volts, D. C. Approvals 145 and 145A, issued to Westinghouse Electric & Manufacturing Co., Nov. 18, 1927.

6. Austin 4 by 5 mine pump; 3-hp. motor, 230 volts, D. C. Approval 157, issued to General Electric Co., Aug. 23, 1928.

7. Austin 5 by 6 mine pump; 5-hp. motor, 230-500 volts, D. C. Approvals 158 and 158A, issued to General Electric Co., Aug. 27, 1928.

8. Deming "Oil-Rite" mine pump; 3-hp. motor, 230 volts, D. C. Approval 161, issued to General Electric Co., Nov. 9, 1928.

9. Deming "Oil-Rite" mine pump; 5-hp. motor, 230 volts, D. C. Approval 163, issued to General Electric Co., Dec. 7, 1928.

10. Scranton 5 by 6 mine pump; 5-hp. motor, 250-500 volts, D. C. Approvals 170 and 170A, issued to the Goodman Manufacturing Co., April 9, 1929.

11. Austin-Brownie 5 by 6 "Perfect Oiler" mine pump; 5-hp. motor, 250-500 volts, D. C. Approvals 199 and 199A, issued to The Brown-Fayro Co., August 18, 1930.

12. Boyts-Porter and Company type No. 6501 mine pump; 5-hp. motor, 230 volts, D. C. Approval 208 issued to Westinghouse Electric & Manufacturing Co., November 29, 1930.

13. Fairmont Mining Machinery Co. 50-gal. mine pump; 5-hp. motor, 230-500 volts, D. C. Approvals 210 and 210A issued to the Fairmont Mining Machinery Co., December 15, 1930.

14. Fairmont Mining Machinery Co., 100-gal. mine pump; 5-hp. motor, 230-500 volts, D. C. Approvals 211 and 211A issued to the Fairmont Mining Machinery Co., December 17, 1930.

15. Fairmont Mining Machinery Co. 100-gal. mine pump; 10-hp. motor, 230-500 volts, D. C. Approvals 213 and 213A issued to the Fairmont Mining Machinery Co., December 29, 1930.

16. Fairmont Mining Machinery Co. 100-gallon mine pump; 7-1/2-hp. motor, 230-500 volts, D. C. Approvals 214 and 214A issued to the Fairmont Mining Machinery Co., January 2, 1931.

17. Fairmont Mining Machinery Co. 50-gallon mine pump; 3-hp. motor, 230-500 volts, D. C. Approvals 215 and 215A issued to the Fairmont Mining Machinery Co., January 2, 1931.

Concrete Mixers

1. Austin-type concrete mixer; 5-hp. motor, 115, 230, and 500 volts, D. C. Approvals 142 and 142A, issued to Westinghouse Electric & Manufacturing Co., Nov. 1, 1927.

Rock-Dusting Machine

1. M.S.A. Co. rock-dusting machine; 5-hp. motor, 230 volts, D. C. Approval 130, issued to Mine Safety Appliances Co., November 5, 1926.

2. Diamond Machine Co. rock-dusting machine; 15-hp. motor, 230 volts, D. C. Approval 131, issued to Diamond Machine Co., December 28, 1926.

3. M.S.A. Co. rock-dusting machine; 15-hp. motor, 230 volts, D. C. Approval 137, issued to Mine Safety Appliances Co., July 2, 1927.

4. Type 125 rock-dusting machine; 12-hp. motor, 250-500 volts, D. C. Approvals 146 and 146A, issued to Mine Safety Appliances Co., Jan. 20, 1928, and April 3, 1928, respectively.

5. Type H rock-dusting machine; 20-hp. motor, 230 volts, D. C. Approval 180, issued to The American Mine Door Co., Oct. 30, 1929.

6. Diamond Machine Co. rock-dusting machine; 12-hp. motor, 95 volts, D. C. Approval 183, issued to Diamond Machine Co., February 4, 1930.

7. Diamond Machine Co. rock-dusting machine; 15-hp. motor, 230 volts, D. C. Approval 195 issued to Diamond Machine Co., July 24, 1930.

8. Type 65 rock-dusting machine; 5-hp. motor, 220 volts, A.C. Approval 206, issued to Mine Safety Appliances Company, November 12, 1930.

PERMISSIBLE ELECTRIC SWITCHES AND JUNCTION BOXES

Approved Under Schedule 4A

1. Enclosed 2-pole fused switch-200 amperes, 250 volts; 100 amperes, 500 volts, D. C. Approvals 400 and 400A, issued to The Ohio Brass Co., June 16, 1928, and Aug. 5, 1925, respectively.

2. Enclosed 2-pole fused switch - 250 amperes, 250 volts; 150 amperes, 500 volts, D. C. Approvals 401 and 401A, issued to the Sullivan Machinery Co., May 11, 1927.

3. Enclosed 3-pole fused switch - 250 amperes, 220 volts; 150 amperes, 440 volts, A. C. Approvals 402 and 402A, issued to the Sullivan Machinery Co., May 11, 1927.

4. Enclosed 3-pole fused switch - 200 amperes, 220 volts; 100 amperes, 440 volts, A. C. Approvals 403 and 403-A, issued to Goodman Manufacturing Co., April 14, 1931.

5. Enclosed 3-pole automatic oil-immersed circuit breaker, type A141B1, 45 amperes, 440 volts, A. C. Approval 404A, issued to Mavor and Coulson (Ltd.), April 16, 1931.

PERMISSIBLE ELECTRIC CAP LAMPS FOR MINERS

Approved Under Schedules 6A and 6B

1. Edison Model "C" lamp. Approval 10, issued to Edison Storage Battery Co., Feb. 24, 1915.

2. Wheat Model C lamp. Approval 17, issued to Koehler Manufacturing Co., Inc., April 30, 1929. (A modification of the Model A lamp approved Sept. 23, 1919).

3. Edison Model H lamp. Approval 18, issued to Edison Storage Battery Co., January 6, 1930. (A modification of the Model E lamp approved Mar. 28, 1923).

4. RM-6 f. d. Ceag lamp. Approval 19, issued to Concordia Electric Co., Aug. 2, 1923.

5. Super-wheat Model X lamp. Approval 20, issued to Koehler Manufacturing Co., Inc., Jan. 30, 1929. (A modification of the original Super-wheat lamp approved April 27, 1926).

6. RM-7 Ceag lamp. Approval 21, issued to Concordia Electric Co., June 18, 1926.

7. Edison Model G lamp. Approval 22, issued to Edison Storage Battery Co., Nov. 16, 1927.

8. RM-8 Ceag lamp. Approval 23, issued to Concordia Electric Co., April 3, 1928.

9. Edison Model J lamp. Approval 24, issued to Edison Storage Battery Co., June 22, 1931.

PERMISSIBLE FLAME SAFETY LAMPS

Approved under Schedules 7, 7A, and 7B

1. Koehler steel frame lamp, flat wick. Approval 201, issued to Koehler Manufacturing Co. (Inc.), Aug. 21, 1915.

2. Koehler Steel frame lamp, round wick. Approval 201A, issued to Koehler Manufacturing Co. (Inc.), July 29, 1918.

3. Koehler Aluminum frame lamp, flat wick. Approval 203, issued to Koehler Manufacturing Co. (Inc.), Feb. 7, 1919.

4. Koehler Aluminum frame lamp, round wick. Approval 203A, issued to Koehler Manufacturing Co. (Inc.), Feb. 7, 1919.

5. Wolf brass frame lamp, round wick. Approval 204, issued to Wolf Safety Lamp Co. of America (Inc.), July 18, 1921.

6. Wolf Aluminum frame lamp, round wick. Approval 205, issued to Wolf Safety Lamp Co. of America (Inc.), April 24, 1924.

7. Wolf Aluminum frame lamp, flat wick. Approval 206, issued to Wolf Safety Lamp Co. of America (Inc.), April 24, 1924.

8. Wolf brass frame lamp, flat wick. Approval 208, issued to Wolf Safety Lamp Co. of America (Inc.), March 14, 1927.

PERMISSIBLE ELECTRIC HAND AND TRIP LAMPS

Approved under Schedule 10A

1. Type EMC-EMCT Ceag hand and trip lamp. Approval 1000, issued to Concordia Electric Co., May 25, 1922.

2. Model E inspection lamp. Approval 1001, issued to Mine Safety Appliances Co., July 28, 1925.

3. Model E signal lamp. Approval 1002, issued to Mine Safety Appliances Co., December 28, 1927.

4. Model E hand lamp. Approval 1003, issued to Mine Safety Appliances Co., Jan. 12, 1928.

5. Super-Wheat hand lamp. Approval 1004, issued to Koehler Manufacturing Co., (Inc.), July 19, 1928.

6. Super-Wheat stevedore lamp. Approval 1005, issued to Koehler Manufacturing Co., (Inc.), August 3, 1928.

PERMISSIBLE ELECTRIC FLASH LAMPS

Approved Under Schedule 11

1. Eveready safety-type flash lamp. Approval 601, issued to National Carbon Co. (Inc.), Oct. 22, 1924.

2. Bond safety-type flash lamp; Approval 602 issued to Bond Electric Corporation, April 6, 1931.

PERMISSIBLE ELECTRIC FLOOD LAMPS
(Portable Type)

Approved Under Schedule 10A

1. Model E and G portable electric floodlight. Approvals 1006E and 1006G, issued to the Mine Safety Appliances Co., February 11, 1931.

PERMISSIBLE METHANE DETECTORS

Approved Under Schedules 7B and 8B

1. Burrell indicating-type detector. Approval 800, issued to Mine Safety Appliances Co., March 10, 1922.

2. Wolf detector, flame type. Approval 207, issued to Wolf Safety Lamp Co. of America (Inc.), Nov. 21, 1924.

3. Martienssen detector. Approval 801, issued to the Gesellschaft für nautische Instrumente, Kiel, Germany, January 9, 1928.

4. "U.C.C." indicating-type detector. Approval 802, issued to Oxweld Acetylene Co., Nov. 27, 1928.

PERMISSIBLE MINE TELEPHONES

Approved Under Schedule 9A

1. Western Electric mine-type telephone. Approval 901, issued to the Western Electric Co. (Inc.), July 16, 1927.

PERMISSIBLE SINGLE-SHOT BLASTING UNITS

Approved Under Schedule 12

1. M.S.A. battery-type blaster. (Edison Model C mine lamp battery with blasting attachment.) Approval 1200, issued to Mine Safety Appliances Co., May 24, 1924.

2. Davis No. 0 magneto-type blaster. Approval 1201, issued to Davis Instrument Manufacturing Co. (Inc.), March 15, 1921.

3. Du Pont Pocket magneto-type blaster. Approval 1202, issued to E. I. Du Pont de Nemours & Co., Aug. 15, 1924.

4. Davis No. 00 magneto-type blaster. Approval 1203, issued to Davis Instrument Manufacturing Co. (Inc.), Oct. 17, 1924.

5. Concordia battery-type blaster. (Concordia type RM-6 f.d. mine lamp battery with blasting attachment.) Approval 1204, issued to the Concordia Electric Co., March 2, 1925.

6. M.S.A. battery-type blaster. (Edison Model E mine lamp battery with blasting attachment.) Approval 1205, issued to the Mine Safety Appliances Co., April 28, 1925.

7. Eveready dry-cell-type blaster. Approval 1206, issued to National Carbon Co. (Inc.), Aug. 20, 1925.

8. Davis No. 000 magneto-type blaster. Approval 1207, issued to Davis Instrument Manufacturing Co. (Inc.), Nov. 18, 1926.

PERMISSIBLE STORAGE BATTERY LOCOMOTIVES AND POWER TRUCKS

Approved Under Schedules 15 and 2C

Gathering Locomotives

1. Whitcomb, E. S. B. flame-proof locomotive. Approval 1500, issued to Geo. D. Whitcomb Co., March 14, 1921.[5]

2. Jeffrey type B.D.M. class 40 locomotive. Approval 1501, issued to The Jeffrey Manufacturing Co., October 11, 1921.

3. Mancha flame-proof "Hercules" locomotive. Approval 1502, issued to the Mancha Storage Battery Locomotive Co., Nov. 13, 1922.

4. Ironton type W.O.G. locomotive. Approval 1503, issued to Ironton Engine Co., March 24, 1923.

5. Mancha Hercules A and AX locomotives. Approval 1505, issued to the Mancha Storage Battery Locomotive Co., April 5, 1924.

6. Jeffrey type B.D.M. Class 25 locomotive. Approval 1507, reissued to the Jeffrey Manufacturing Co., Aug. 20, 1925.

7. Goodman Type 10-30 locomotive. Approval 1508, issued to the Goodman Manufacturing Co., March 21, 1925.

8. Goodman Type 8-30 locomotive. Approval 1509, issued to the Goodman Manufacturing Co., September 25, 1925.

9. Mancha Standard A, AN, and AX locomotives. Approval 1511, issued to Mancha Storage Battery Locomotive Co., Nov. 10, 1925.

10. Westinghouse-Baldwin locomotive. Approval 1512, issued to Westinghouse Electric & Manufacturing Co., Nov. 11, 1925.

11. General Electric type LSBE-2C6-C9 locomotive. Approval 1513, issued to General Electric Co., February 25, 1926.

12. Jeffrey Type B.D.M. 25 Form H. Locomotive. Approval 1516, issued to The Jeffrey Manufacturing Co., December 28, 1926.

13. Atlas Type B locomotive. Approval 1517, issued to Atlas Car & Manufacturing Co., Feb. 10, 1927.

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5 This approval was rescinded for cause on May 12, 1931, without prejudice to the successor company, The Whitcomb Locomotive Co.

14. Mancha 2-motor locomotive; Approval 1520, issued to the Mancha Storage Battery Locomotive Co., May 27, 1929.

15. Jeffrey type DM-15 locomotive. Approval 1521, issued to The Jeffrey Manufacturing Co., June 13, 1930.

16. General Electric type LSB-2C5-F-324 locomotive. Approval No. 1522, issued to the General Electric Co., September 12, 1930.

17. "Midget" locomotive. Approval No. 1523, issued to the Westinghouse Electric & Manufacturing Co., December 19, 1930.

Main-Line Haulage Locomotive

1. Jeffrey Type B.D.M. Class 30 main-line haulage locomotive. Approval 1510, issued to Jeffrey Manufacturing Co., Oct. 12, 1925.

Power Trucks

1. Mancha power tank. Approval 1506, issued to Mancha Storage Battery Locomotive Co., May 5, 1924.

2. Mancha power tank and gathering locomotive, Approval 1505A, issued to Mancha Storage Battery Locomotive Co., June 21, 1926.

3. Jeffrey power truck and main-line haulage locomotive. Approval 1510-C, issued to The Jeffrey Manufacturing Co., Dec. 31, 1926.

4. Mancha nonpropelled power truck, Approval 1514, issued to Mancha Storage Battery Locomotive Co., December 18, 1926.

5. Jeffrey power truck. Approval 1515, issued to The Jeffrey Manufacturing Co., December 28, 1926.

6. Westinghouse power truck. Approval 1512-C, issued to Westinghouse Electric and Manufacturing Co., Sept. 13, 1928.

7. General Electric type LSBE-2C6-C12 power truck and gathering locomotive. Approval 1519-C, issued to the General Electric Co., April 6, 1929.

Tandem Locomotives

1. Jeffrey tandem locomotive. Approval 1518, issued to the Jeffrey Manufacturing Co., Nov. 21, 1927.

APPENDIX

Specially Recommended Trailing Cables
(Listed according to their Bureau of Mines symbol numbers)

1. BM-1 "Hazacord"[6] No. 2 twin cable (19 x 7 stranding).
2. BM-2 "Hazacord" No. 3 twin cable (19 x 7 stranding).
3. BM-3 "Hazacord" No. 4 twin cable (19 x 7 stranding).
4. BM-4 Rome "Super-Service"[7] No. 3 twin cable (7 x 19 stranding).
5. BM-5 Rome "Super-Service" No. 2 twin cable (7 x 19 stranding).
6. BM-6 Rome "Super-Service" No. 4 twin cable (7 x 19 stranding).
7. BM-7 Rome "Super-Service" No. 4 twin cable (7 x 7 stranding).
8. BM-8 Okocord[8] No. 2 twin cable (19 x 7 stranding).
9. BM-9 Tirex[9] No. 2 twin cable (7 x 19 stranding).
10. BM-10 Tirex No. 3 twin cable (7 x 19 stranding).
11. BM-11 Tirex No. 4 twin cable (7 x 19 stranding).

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6 Manufactured by the Hazard Insulated Wire Works, Division of the Okonite Co.
7 Manufactured by the General Cable Corporation.
8 Manufactured by the Okonite Co.
9 Manufactured by the Simplex Wire & Cable Co.

I. C. 6539

November, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

TOURMALINE

BY

I. Aitkens

I.C.6539.
November, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

TOURMALINE[1]

By I. Aitkens[2]

CONTENTS

INTRODUCTION

Tourmalines are among the most beautiful of all the semiprecious stones, and are unsurpassed even by corundum in variety of hue. The stones most suited to jewelry purposes are those comparatively free from iron. One reason for the fact that the tourmaline has advanced rapidly in public favor in this country is that it is found in California in great profusion and of extraordinary quality. The superb green and especially the pink crystals found at Pala and Mesa Grande in San Diego County, Calif., are deemed superior to tourmalines from all other localities.

DESCRIPTION AND PROPERTIES

Tourmalines are borosilicates of very complex composition. Authorities differ as to the correctness of the formula, but it is often written $R_{18}B_2(Si\ O_5)_4$, in which R may include lithium and other alkalies, magnesium, calcium, manganese, iron, and aluminum. Fluorine likewise may be present.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6539."

2 - Rare metals and nonmetals division, U. S. Bureau of Mines.

The mineral is commonly found in well-developed, prismatic, vertically grooved or striated crystals of the hexagonal system. The crystals usually exhibit a rounded triangular outline, and different crystal forms are present at the opposite ends. This polar character is further evidenced by the electrical properties of tourmaline.

Tourmaline has no well-defined cleavage, and its fracture is from subconchoidal to uneven. It is optically negative and has a strong birefringence (0.020). Its mean index of refraction is near 1.63, and the dispersion is 0.016. The most prominent character of this mineral is its strong dichroism, and therefore stones are usually cut so that both of the dichroic colors may be seen as the stone is turned in various directions.

The hardness is about the same as quartz, from 7 to 7.5 on Moh's scale, but it is slightly heavier, the specific gravity ranging from 2.9 to 3.2. Tourmaline has a vitreous luster, and while the light colored stones are more or less transparent, the black and brown varieties are generally opaque. However, usually the tourmaline that is rich in alkalies is transparent and possesses most attractive colors. The varieties used as gems are achroite, colorless; Brazilian emerald, green; Brazilian peridot, yellowish green; Brazilian sapphire, blue; indicolite, dark blue; peridot of Ceylon, honey-yellow; rubellite, rose-red or pink; and siberite, violet. The iron tourmaline or schorl is black, and the magnesium tourmaline is brown.

In the alkali tourmalines the colors are frequently arranged in regular zones or bands. These zones may be horizontal, running across the crystals; or the colors may be arranged in vertical, somewhat cylindrical zones, parallel to the prism faces of the crystal. The typical arrangements of the colors, however, are as follows: Green at one end and red at the other, with a narrow colorless zone between; green followed by yellow, red, and green again; or, perhaps crimson or green tipped with black. In Brazilian stones the core is generally red, with a marginal zone of green and an intermediate colorless band; the southern California stones are usually green inside and red outside.

The following is a list of gem and trade names of the tourmaline mineral group:

Gem Name	Common Name or Description
Achroite	Colorless or white tourmaline.
Andalusite	Silicate of aluminum; also trade name for brown tourmaline.
Aphrizite	Black tourmaline.
Brazilian emerald	Green tourmaline.
Brazilian peridot	Yellow-green tourmaline.
Brazilian sapphire	Light-blue or greenish topaz; also blue tourmaline.
Ceylon chrysolite	Yellowish green or greenish yellow tourmaline.
Ceylon peridot	Honey-yellow or yellowish green tourmaline.
Dravite	Brown tourmaline.
Emeralite	Green and bluish green tourmaline from San Diego County, Calif.

Gem Name	Common Name or Description
Indicolite	Blue tourmaline.
Peridot of Ceylon	Same as Ceylon peridot, honey-yellow tourmaline.
Precious schorl	Tourmaline.
Rubellite	Pink and red tourmaline.
Schorl	Black tourmaline.
Siberian ruby	Red tourmaline.
Siberite	Violet-red tourmaline.

USES

The fact that tourmaline is soft and consequently not very durable discourages its use where it is likely to encounter hard wear. Although well suited for pendants, brooches, and like articles, the stone must be carefully protected by the setting when mounted in rings. It is admirably fitted for use in any article of jewelry in which a large central stone is required. In the cutting of gem tourmaline various styles are used. Occasionally it is sufficiently fibrous in structure to display, when cut "en cabochon," a pronounced chatoyancy (play of colors), but, like all colored stones, it is generally brilliant-cut in front and step-cut at the back.

Tourmaline is also used in the manufacture of tourmaline tongs and in measuring the intensity of radium emanations.

Ordinary black tourmaline usually has no commercial value but the United States Bureau of Mines has received one or two inquiries for small lots of a few pounds for experimental purposes, probably in connection with radio or other electrical work.

IDENTIFICATION

Tourmaline is recognized chiefly by the characteristic rounded triangular cross section of its crystals. It is easily distinguished from topaz, because the latter is about 30 per cent heavier than either tourmaline or quartz. Its range of refractivity, on the other hand, overlaps that of topaz although the latter has a much smaller double refraction by which it may be distinguished readily. Unmounted stones are distinguished even more easily, because tourmaline floats in methylene iodide, whereas topaz sinks.

Another distinguishing feature of tourmaline is that it is strongly pyroelectric, and when cooling after being heated it will develop positive electricity at one end of the crystal and negative at the other. It is also strongly dichroic, and light traversing the crystal in one direction may be quite a different color or shade of color from that traversing the crystal in a direction at right angles to the first.

SUBSTITUTES

Tourmalines are remarkable for certain optical properties which they possess and which render them practically incapable of being successfully imitated. However, there is little or no commercial advantage in manufacturing artificial tourmalines, due to their relatively low value and abundance in many localities.

HISTORY

Although tourmalines were no doubt known to the ancients, it was a long time before the present name was adopted. The name of the species is derived from the ancient Singhalese word tumali, and not turamali, which is applied by the natives of Ceylon to the yellow zircon of that island.

As the story goes, one warm summer's day early in the eighteenth century, some children of Amsterdam were playing with stones brought home by Dutch navigators and noticed the strange attractive powers developed by the exposure of the stones to the heat of the sun. Their parents, summoned to view the phenomenon, were likewise astonished to find that these stones could attract or repel, with a decided force, ashes, straws, and other light substances. The Dutchmen proceeded to call them aschentreckers, or "ash-drawers." Schorl, a subsequent title, afterwards gave way in favor of tourmaline.

MODE OF OCCURRENCE

Tourmaline occurs in compact or disseminated masses in rock and as loose, rounded crystals in secondary deposits. It is a common accessory mineral in pegmatites associated with granite intrusions, and also in metamorphic rocks, such as gneisses, schists, and crystalline limestones. The light-colored gem varieties are found chiefly in pegmatites. The minerals ordinarily found associated with tourmaline are orthoclase, albite, quartz, muscovite, lepidolite, beryl, apatite, fluorite, etc.

MINING METHODS

Generally, in mining for tourmalines, the rock is drilled by hand and the blasting is done with black powder in preference to dynamite, so as to minimize the risk of shattering the valuable gem material. Where a considerable thickness of schist overlies the pocket-bearing layer, tunneling is the most economical method of working. After the rock is broken, the tourmaline crystals are removed carefully by hand.

DOMESTIC PRODUCTION

The value of the production of tourmaline in the United States at one time was of considerable importance. During the period of 1902-1910, inclusive, the domestic production exceeded $30,000 annually and in 1909 the value of the output reached $133,192. The high mark reached during this peak year was attributed to trade with the Chinese merchants in pink tourmalines from California. Although subsequent years showed a decided decline, in 1916 the value of production reached the $50,000 mark. Production statistics for the United States from 1898 to 1921, when the United States Geological Survey discontinued the canvass of producers are given in the following table. Later figures are not available.

Table 1.- Value of tourmalines produced in the United States, 1898-1921[1]

Year	Value	Year	Value	Year	Value
1898	$4,000	1906	$72,500	1914	$7,980
1899	2,000	1907	84,120	1915	10,969
1900	3,500	1908	90,000	1916	50,807
1901	15,000	1909	133,192	1917	12,452
1902	30,000	1910	46,500	1918	6,206
1903	45,000	1911	16,445	1919	17,700
1904	40,000	1912	28,200	1920	4,869
1905	50,000	1913	7,630	1921	1,450

1/ Compiled from Annual Chapters of the Mineral Resources of the United States, Pt. II.

DEPOSITS IN THE UNITED STATES

In the United States, California and Main have been the chief tourmaline-producing States. These widely separated regions are both famous for their beautiful colored tourmalines. In Maine, are the deposits at Mount Mica, Mount Apatite, and Mount Rubellite near the towns of Paris, Auburn, and Hebron, respectively; while in California, the principal deposits are found in the Mesa Grande district in San Diego County.

Other localities in this country are Haddam, Conn.; Chesterfield and Goshen, Mass.; Gouverneur, De Kalb, and Pierrepont, N. Y.; and Chester, Pa. Deposits of lesser importance have been found also in several other States.

Arizona

Upon examination of the quicksilver deposits of Arizona, cinnabar was found closely associated with tourmaline. As cinnabar in so many of its occurrences has been deposited near the surface and at relatively low temperatures, and tourmaline is a mineral characteristic of pegmatite dikes and of veins belonging to the deep zone and formed at high temperatures, the occurrence of these two minerals in association was considered worthy of record. Lausen[3] has described in considerable detail the occurrence of tourmaline in the cinnabar veins of the Mazatzal Mountains of Arizona.

California

California tourmalines were known as early as 1872, when the Indians and cowboys gathered the crystals from the Mesa Grande region in San Diego County. After commercial exploitation took place, it was found that the tourmaline-bearing belt lies in San Diego and Riverside Counties and includes Mesa

3 - Lausen, Carl, Tourmaline-bearing Cinnabar Veins of the Mazatzal Mountains, Arizona: Econ. Geol., Dec., 1926, pp. 782-791.

Grande, Pala, and Coahuila Mountain as the principal localities. Mesa Grande is by far the most interesting locality, being noted for pink and variegated tourmalines and for having yielded the large and magnificent groups now seen in some of the museums.

According to Foshag,[4] no other known locality produces tourmalines equal to the California specimens. While the California stones are inferior to the Maine (Mt. Mica) stones in beauty and color, they surpass them in size and crystal perfection. Many of the larger crystals exceed $5\frac{1}{2}$ inches in length and $3\frac{1}{2}$ inches in thickness, and a large number have been found with the crystal faces developed at both ends.

The California tourmalines rarely show concentric bands, although a few are green in the center and pink outside. The more striking forms are longitudinally banded. For instance, the wonderfully beautiful crystals from Pala are one-half deep pink with the remainder clear green. The larger crystals from Mesa Grande usually show a uniform pink capped with green, or sections of green, pink, and salmon.

In the Mesa Grande region in San Diego County, Calif., the mineral occurs also in pegmatitic rocks associated with spodumene, commercially known as kunzite, and occasional gem minerals of other varieties.

These different localities have been described by Kunz.[5]

At one time the trade with Chinese merchants in pink tourmaline from California had grown to quite a considerable industry. The estimated value of purchases by the Chinese during 1909 alone was close to $100,000, and during the summer of 1910, the Himalaya Mining Co., sold to these merchants a large consignment which consisted of 358,500 carats of pink tourmaline crystals ranging in size from 100 to 1,000 carats.

Maine

Mount Mica, which is situated about $1\frac{1}{2}$ miles east of Paris, Me., is undoubtedly one of the most famous mineral localities in the United States. Since 1820, it has been known to mineralogists all over the world because of the size and beauty of its tourmaline crystals. A good description of this locality has been given by E. S. Bastin,[6] and the earlier history in detail was written by A. C. Hamlin.[7]

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4 - Foshag, W. F., Gems and Gem Minerals: Smithsonian Sci. Ser., vol. 3, pt. 2, 1929, pp. 238-243.

5 - Kunz, Geo. F., Gems, Jewelers' Materials and Ornamental Stones of California: California State Min. Bur. Bull. 37, San Francisco, June, 1905, pp. 54-63.

6 - Bastin, E. S., Geology of the Pegmatites and Associated Rocks of Maine: U. S. Geol. Survey Bull. 445, 1911, pp. 81-93.

7 - Hamlin, A. C., The History of Mount Mica: Bangor, Me., 1895, 72 pp.

A large number of unusually fine gems have been cut from the tourmalines of Mount Mica. Green is the predominant color of the gem tourmalines, but this district has yielded fine rubellite, indicolite, and achroite gems of varying hues. The colorless achroite crystals grade into delicate pink, green, and blue; the indicolites are greenish blue and nearly sapphire blue; and the rubellite runs from pale to deep pink and nearly ruby-red. The combinations of these various colors in a single crystal, have furnished some very remarkable specimens for mineral collections now scattered through many parts of the world.

The Mount Apatite district also has yielded some very fine gem crystals. A number of quarries were opened for feldspar and tourmaline on Mount Apatite about 3½ miles west of Auburn, Me., from which pockets were discovered which yielded some very fine colored tourmalines. One quarry on the west side was worked by W. R. Wade,[8] and has been described along with other gem deposits of Maine. Besides those found and described by Wade, other pockets with purple apatite have been unearthed, but such crystals have been sold chiefly for cabinet specimens, because owing to the comparative softness of apatite they would not wear well if cut into gems.

FOREIGN LOCALITIES

Tourmalines[9] and especially rubellites (which are found in all colors) occur in nearly every country of the globe but rarely in sufficient quantities to mine.

Magnificent crystals of pink, blue, and green have been found in the neighborhood of Ekaterinburg, principally at Mursinka, in the Urals, Russia, and fine rubellite has come from the Urula River, and other spots near Nertschinsk, Transbaikal, Asiatic Russia. Elba, the Italian island on which Napoleon was exiled, produces pink, yellowish, and green stones, frequently particolored, and sometimes the crystals are blackened at the top and are called "nigger heads." Ceylon supplies small yellow stones which are often confused with zircon of a similar color. Rubellite accompanies the ruby of Ava, Burma, and rivals the ruby itself in color.

In Canada, amber-colored crystals are found at Fitzroy, Ontario; transparent brown at Hunterstown, Quebec; black at Bathurst and Elmsley, Ontario and St. Jerome, Quebec; and magnificent green-yellow crystals occur in the limestones at Great Calumet Island. Fine tourmalines in a great variety of colors, and frequently zoned are found at many localities in Madagascar. These stones compare favorably in beauty, if not in size, with any found elsewhere. Tourmalines are found in Siberia and Sweden also, and to a lesser extent in many localities; but, Brazil is undoubtedly the greatest tourmaline country.

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8 - Wade, W. R., The Gem-bearing Pegmatites of Western Maine: Eng. and Min. Jour., vol. 87, 1909, pp. 1127-1129.

9 - Pan American Union Bulletin, The Semiprecious Stones of Brazil: Washington, April, 1925, pp. 371-377.

Brazil

To-day Brazil is by far the most important source of tourmalines. Practically all the tourmalines mined in Brazil are from the state of Minas Geraes near the border of Espirito Santo, and the city of Lajao is the most important mining center. Rubellite is the most common variety and exists in all colors.

In the Brazilian stones the core is generally red, bounded by white, with green on the exterior. However, beautiful crystals of green and red, often diversely colored, come from various parts of the State of Minas Geraes, Brazil. According to A. S. Atkinson,[10] as far back as 1909 tourmaline mining has been one of the most important industries in Brazil. In view of the fact that precious stones are so profuse in Brazil and semi-precious stones are found in such large quantities, it is not surprising to note that this country ranks first among the tourmaline sources of the world.

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10 - Atkinson, A. S., Mining for Gems in Brazil: Eng. and Min. Jour., vol. 87, June 19, 1909, p. 1234.

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Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINE EXPLOSIONS AND FIRES IN THE UNITED STATES DURING THE FISCAL YEAR ENDING JUNE 30, 1931

BY

D. Harrington

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINE EXPLOSIONS AND FIRES IN THE UNITED STATES DURING THE FISCAL YEAR ENDING JUNE 30, 1931[1]

By D. Harrington[2]

Approximately 2,000 persons are killed annually in the coal mines of the United States. There is no question that this number is at least double and probably treble what it should be and by all means the least excusable of the fatalities are those caused by explosions and fires. Fatalities from explosions and fires in coal mines are almost invariably caused by carelessness or negligence. Table 1 indicates that the fiscal year ending June 30, 1931, was by no means a satisfactory one as regards either the number of explosions or the number of persons killed by them. This tabulation lists all ignitions to which the attention of the personnel of the safety division was called and includes not only major disasters, but also explosions in which fewer than five were killed, as well as several explosions from which no fatalities resulted.

It will be noticed that last year there were 26 explosions in mines in 10 States of the United States, resulting in 217 deaths. In the previous year there were 34 explosions with a death list of 199, and in the year before there were 38 explosions with a death list of 139. Hence, the past fiscal year had distinctly a poorer record of fatalities from mine explosions than either of the two previous years.

Of the 26 explosions in the past fiscal year, Pennsylvania had 7, West Virginia 4, New Mexico and Ohio 3 each, Alabama, Oklahoma and Washington 2 each, and California, Illinois and Indiana 1 each; the one in California was in a tunnel being driven for water purposes but giving off explosive gas (methane). In fatalities Ohio had the worst record with 88, Oklahoma came next with 45, then Indiana with 28, and Pennsylvania with 23; although Alabama had 2 ignitions and Washington 1, they caused no fatalities. Utah, Colorado, and Kentucky had no explosions reported by the safety division of the Bureau of Mines during the past fiscal year, but were on the list for the previous year, though in the case of Colorado there were no fatalities. On the other hand, Indiana and New Mexico, neither of which was charged with an explosion in the two previous years, were on the list for the past fiscal year; New Mexico had the bad record of three explosions, though fortunately with but a total of eight fatalities. Ohio reported 3 explosions to the safety division of the Bureau of Mines in the past fiscal year, 2 in the previous year, and 3 in the year before that, or 8 in the past three fiscal years with 92 fatalities; all of these eight explosions were in open-light mines.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6540."

2 Chief engineer, safety division, U. S. Bureau of Mines.

Table 1.- Summary of mine explosions by States, July 1, 1930 to June 30, 1931

	Lighting				Fatalities by ignition causes									
	Open	Closed	Unknown	Total mine explosions	Electricity		Open lights or smoking		Explosives		Unknown		Total	
					Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions
Alabama	2	0	-	2	0	0	0	2	-	-	-	-	0	2
California[1]..	-	1	-	1	12	1	-	-	-	-	-	-	12	1
Illinois	-	1	-	1	2	1	-	-	-	-	-	-	2	1
Indiana	1	-	-	1	-	-	28	1	-	-	-	-	28	1
New Mexico ..	1	2	-	3	7	2	0	0	1	1	-	-	8	3
Ohio	3	-	-	3	82	1	0	0	6	2	-	-	88	3
Oklahoma	-	2	-	2	30	1	15	1	0	0	-	-	45	2
Pennsylvania[2]	3	5	-	7	0	0	12	4	11	3	-	-	23	7
Washington ..	1	1	-	2	-	-	0	2	0	0	-	-	0	2
West Virginia	1	3	-	4	10	3	1	1	-	-	0	0	11	4
Total United States fiscal year ending June 30, 1931	12	15	0	26	143	9	56	11	18	6	0	0	217	26
Total United States fiscal year ending June 30, 1930[3]	8	25	1	34	154	18	21	8	24	6	0	2	199	34
Total United States fiscal year ending June 30, 1929[4]	18	19	1	38	93	18	14	11	26	8	6	1	139	38

1 These explosions occurred in tunnels where methane was being given off and where incandescent lighting was in use.

2 One mine operated with mixed lights.

3 See Information Circular 6419, January, 1931.

4 See Information Circular 6178, October, 1929.

Table 1 shows that 143 out of the 217 fatalities in explosions in the past fiscal year were from explosions with ignition by electricity, again demonstrating that electricity is one of the most dangerous agencies in our mines; 56 of the 217 fatalities were from explosions caused by open lights--far more than the 21 fatalities from that cause in the previous fiscal year and the 14 in the year before; 18 of the 217 deaths from last year's explosions were from explosions started by explosives or blasting, a decrease from the 24 or the 26 from this cause in the two previous years.

EXPLOSIONS CAUSED BY ELECTRICITY

Table 1 shows that but 9 of the 26 explosions for the past fiscal year were caused by electrical ignition, though 143 of the 217 deaths were due to those 9 explosions of electrical origin. Table 2 below lists the electrical causes of explosions not only for the fiscal year ended June 30, 1931, but also for the two previous fiscal years.

Table 2.- Electrical causes of explosions

Causes	Number of ignitions			Total, three years
	1928-1929[1]	1929-1930[2]	1930-1931	
Trolley or cable-reel locomotive..	5	5	1	11
Nonpermissible mining machine	5	5	3/ 3	13
Nonpermissible motor	4	3		7
Nonpermissible shot-firing battery	-	4/ 2	4/ 5/ 3	2
Cable nips or blow-out	-	2		2
Nonpermissible storage-battery locomotive	1	1	6/ 2	4
Shot-firing line	-	1		1
Faulty wiring	2	1		3
Trolley wire	1	-	7/ 2	3
Electric arc	2	-	8/ 1	
Total ignitions	20	18	9	49

1 See Information Circular 6178.
2 See Information Circular 6419.
3 Thirty-four killed.
4 Not included in totals; may have been from overcharged shots and therefore are listed as explosives accidents in Table 1.
5 Ten killed.
6 Twenty killed.
7 Eighty-four killed.
8 Five killed.

It will be noticed that 49 explosions in the last three fiscal years were started by electricity (resultant deaths from these 49 explosions amounted to 390, see Table 1); 11 of these 49 explosions were caused by trolley or cable-reel locomotives, 4 by nonpermissible storage-battery locomotives, and 3 by trolley wire, or 18 out of the 49 by nonpermissible locomotives or wiring for locomotives; surely this in itself is a severe indictment of the use of such

equipment in any coal mine; in the past fiscal year one explosion with 82 fatalities was caused by an arc from a fall of roof material on a trolley wire. Thirteen of the 49 explosions of electrical origin in the past three years were due to nonpermissible mining (coal-cutting) machines; there isn't in the record even one explosion caused by a permissible mining machine, and since efficient permissible mining machines are available to all, there doesn't appear to be any good reason why nonpermissible mining machines should be used in coal mines. Of the 49 explosions of electrical origin during the past three fiscal years, 37 or 75 per cent were due to nonpermissible electrical equipment (locomotives, mining machines, motors, and shot-firing batteries); when it is remembered that these 49 explosions caused 390 fatalities, not one due to permissible electrical equipment, one marvels that nonpermissible electrical equipment is allowed in coal mines; there is at any rate every warrant for the exclusion of nonpermissible equipment from any mine or part of mine which gives off methane or has coal-dust undiluted by thorough rock-dusting.

CAUSES OF EXPLOSION IGNITIONS FOR THE PAST FOUR FISCAL YEARS

Table 3 lists the explosions by number and cause of ignitions during the past four fiscal years as reported by the field personnel of the safety division of the United States Bureau of Mines.

Table 3.- Comparison of explosion ignition causes, fiscal years 1927-28 to 1930-31, inclusive

Years	Electricity		Open lights or smoking		Explosives		Unknown		Total	
	No.	Per cent	No.	Per cent	No.	Per cent	No.	Per cent	No.	Per cent
1927-28	14	45.1	12	38.7	3	9.7	2	6.5	31	100.00
1928-29	18	47.4	11	29.0	8	21.0	1	2.6	38	100.00
1929-30	18	52.5	8	23.5	6	18.0	2	6.0	34	100.00
1930-31	9	34.6	11	42.3	6	23.1	0	0.0	26	100.00
Total	59	45.8	42	32.5	23	17.8	5	3.9	129	100.00

It will be seen that in the last fiscal year 34.6 per cent of the explosions reported to the safety division of the Bureau of Mines were started by electricity, and for the past four years 59 out of 129 explosions, or 45.8 per cent, were caused by electricity; last year 42.3 per cent of the explosions were caused by open lights or smoking, while for the past four years 42 out of 129 explosions or 32.5 per cent were caused by open lights or smoking. Last year 23.1 per cent of the explosions were started by explosives or blasting, and for the 4-year period 23 of the 129 explosions or 17.8 per cent were due to blasting. It is distinctly disquieting to realize that notwithstanding all that is known about prevention of explosions, those disasters continue in our mines and have occurred at an average of 32 per year for the past four years.

FATALITIES FROM VARIOUS EXPLOSION IGNITION CAUSES

Table 4 shows that in the mines of the United States during the past four years 897 persons have been killed in explosions to which attention of the safety division personnel has been called. The fiscal year 1927-28 had the worst record with its 342 fatalities from explosions, and the past fiscal year (1930-31) was second with 217 killed; the record of the latter year, however, was but little worse than that of the preceding year (1929-30) when the explosion fatalities in mines reached 199.

Table 4.- Comparison of explosion fatalities by ignition causes, fiscal years 1927-28 to 1930-31, inclusive

Years	Electricity		Open lights		Explosives		Unknown		Total	
	Number killed	Per cent	Number killed	Per cent	Number killed	Per cent	Number killed	Per cent	Number killed	Per cent
1927-28	282	82.5	49	14.3	9	2.6	2	0.6	342	100.00
1928-29	93	66.9	14	10.1	26	18.7	6	4.3	139	100.00
1929-30	154	77.5	21	10.5	24	12.0	0	0.0	199	100.00
1930-31	143	65.9	56	25.8	18	8.3	0	0.0	217	100.00
Total (4 years)	672	74.8	140	15.6	77	8.7	8	0.9	897	100.00

In the past fiscal year 143 fatalities occurred in explosions started by electricity, these deaths amounted to 65.9 per cent of the 217 killed in all explosions; the fact that during the past four years 672 persons have been killed in our mines in explosions started by electricity is one which should cause every right-minded mining man to take prompt decisive action toward safeguarding his employees from possible disasters because of defective electrical equipment or misuse of electrical equipment in mines. During the last four years 74.8 or essentially 75 per cent of the fatalities from explosions in our mines have been due to explosions started by electricity!

In the past fiscal year open lights caused explosions which accounted for 25.8 per cent of the explosion fatalities, this being more than double the percentage for either of the two previous years and nearly double the rate for the fiscal year ending June 30, 1928; moreover, the open light was not charged with ignition of an explosion with heavy loss of life in an open-light mine, though some people believe that this disaster might well be charged to open lights; there is no question that the open light is ever a menace to the safety of all the workers of any mine which uses it.

In the year ended June 30, 1931, blasting or the use of explosives caused explosions with 18 fatalities, or 8.3 per cent of the total for the year, this percentage being essentially the same as the 8.7 per cent for the 4-year period ending June 30, 1931. Precautions taken in the use of explosives in mines are being extended year by year, and there is good reason to hope that mine explosions with heavy loss of life due to blasting will

gradually disappear--unless some of the relaxations now being taken in blasting for mechanized loading should cause some bad disasters, and this is by no means improbable.

The fact that in the last four fiscal years 897 persons were killed in explosions in our mines is a sad commentary on the efficiency of our mining men, as there is absolutely no question that mine explosions are preventable and that practically all of them are the result of carelessness. There is, however, a silver lining to this cloud; during the last five months of the fiscal year ended June 30, 1931, there were but 10 fatalities in coal mines of the United States from explosions; moreover the bituminous mines of Pennsylvania with upwards of 130,000 employees have not had a major disaster (one with five or more killed) since March 21, 1929, or for nearly 2½ years; this shows what can be done.

Table 5.- Comparison of lighting in mines where explosions occurred, fiscal years 1927-28, to 1930-31, inclusive

Year	Open lights		Closed lights		Lighting Unknown		Total	
	Number of explosions	Per cent	Number of explosions	Per cent	Number of explosions	Per cent	Number of explosions	Per cent
1927-28	14	45.1	15	48.4	2	6.5	31	100.00
1928-29	18	47.4	19	50.0	1	2.6	38	100.00
1929-30	8	23.5	25	73.5	1	3.0	34	100.00
1930-31	12	44.4	15	55.6	0	0.0	1/ 27	100.00
	52	40.0	74	56.9	4	3.1	1/130	100.00

1 One mine had mixed (open as well as closed) lighting.

Table 5 shows that in the past fiscal year 12 out of 27 explosions, or 44.4 per cent, were in open-light mines, while 15 or 55.6 per cent were in closed-light mines; for the past four fiscal years 52 of 130 explosions, or 40 per cent, were in open-light mines and 74 or 56.9 per cent were in closed-light mines. The significance of this is subject to some debate, but one thing must be kept in mind very definitely: Open lights have caused numerous explosions (in fact, nearly all of them) in open-light mines (and a few in so-called closed-light mines), but no permissible electric cap lamp has ever caused an explosion in any mine whether on an open or a closed light basis. The closed-light mines are usually those known to be dangerous because of gas or dust or both, hence have potential hazards much in excess of those in which open lights are allowed ordinarily; however, there is no question that where closed lights are used mining companies in many instances are prone to rely too much on the safety features of closed lights and to take far too many chances with nonpermissible electrical equipment, to use electrical equipment under dangerous conditions as to gas or dust or both, and only too frequently ventilation precautions are relaxed, apparently with the idea that the closed lights will prevent trouble from those relaxations or deficiencies in safety practice. Open lights ought not to be allowed in any mine,

particularly any coal mine, as there will continue to be fires and explosions in mines as long as open lights or any type of open flame is allowed underground; there will also continue to be fires and explosions in closed-light mines as long as unsafe electrical equipment, unsafe practices with electricity, unsafe blasting materials and practices, and various kinds of carelessness with ventilation are allowed in closed light mines. In other words, while open lights are at all times and under all circumstances dangerous in mines, on the other hand closed lights in themselves are by no means a cure-all for the ills of underground workers or workings.

Table 6 lists the mine fires to which the personnel of the safety division were called or with which they were in contact during the fiscal year ending June 30, 1931. It will be noticed that there were 24 fires with 23 fatalities in 14 States. In the previous fiscal year the safety division also sent men to 24 fires in 14 States; fatalities, however, were but five. In the past fiscal year 12 of the fires were in open-light mines as against but 7 in open-light mines in the previous year; last year 7 fires were in closed-light mines as against 8 in the previous year; the lighting system was unknown in five mines which had fires in the past fiscal year, and in nine mines the previous year.

Electricity caused 7 fires but no fatalities from those fires in the past fiscal year, as against 9 fires with 2 fatalities in the previous year; 3 fires with 5 fatalities were caused by open lights or smoking in the past year as against 6 fires and 3 fatalities in the previous year; explosives caused 5 fires with no fatalities last year and 3 fires with no fatalities in the previous year (it is probable that several hundred coal-mine fires were caused by explosives, but in general they were not called to the attention of the safety division of the Bureau of Mines); last year 9 fires due to miscellaneous or unknown causes with 18 deaths were attended by the safety division personnel; one of these, causing 11 deaths in a city tunnel being driven for sewer purposes, was probably caused by smoking, though it may have been due to defective wiring. Three of the 24 fires listed in Table 6 were not in coal mines; one with 11 fatalities was in a tunnel being driven in a city, and 2 with 2 fatalities were at metal mines.

In addition to 26 explosions with 217 fatalities, and 24 fires with 23 fatalities, the personnel of the safety division had fairly close contact during the past fiscal year with 45 other miscellaneous accidents with 75 fatalities, a total of 95 accidents of all kinds with fatality total of 315. In the previous fiscal year safety division personnel were in contact with 34 explosions with 199 fatalities, 24 fires with 5 fatalities, and 29 miscellaneous accidents with 72 fatalities, a total of 87 accidents of all kinds with 276 fatalities.

Table 6.- Summary of mine fires by States, July 1, 1930 to June 30, 1931

	Lights				Fatalities by ignition causes									
					Electricity		Open lights or smoking		Explosives		Miscellaneous		Total	
	Open	Closed	Unknown	Total	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions
Alabama	-	-	1	1	-	-	-	-	-	-	0	1	0	1
California ..	1	1	-	2	0	0	5	1	-	-	0	1	5	2
Colorado	-	-	1	1	-	-	-	-	-	-	0	1	0	1
Illinois	1	-	1	2	0	1	-	-	-	-	11	1	11	2
Indiana	2	-	-	2	-	-	-	-	0	2	0	-	0	2
Kentucky	2	0	-	2	0	1	0	0	0	1	-	-	0	2
Ohio	2	0	-	2	0	1	0	0	-	-	4	1	4	2
Oklahoma	0	2	0	2	0	1	0	0	0	1	-	-	0	2
Pennsylvania.	1	0	-	1	0	1	0	0	0	0	-	-	0	1
South Dakota.	1	0	0	1	0	0	0	0	0	0	2	1	2	1
Utah	0	1	0	1	0	0	-	-	-	-	0	1	0	1
Washington...	-	2	-	2	0	1	-	-	-	-	0	1	0	2
West Virginia	1	1	2	4	0	1	0	1	0	1	1	1	1	4
Wyoming	1	-	0	1	-	-	0	1	-	-	0	0	0	1
Total fiscal year ending June 30, 1931	12	7	5	24	0	7	5	3	0	5	18	9	23	24
Total fiscal year ending June 30, 1930[1]	7	8	9	24	2	9	3	6	0	3	0	6	5	24

1 See Information Circular 6419, issued January, 1931.

CONCLUSION

Although it is disheartening to realize that more men were killed in mine explosions during the fiscal year ending June 30, 1931 than in either of the two previous years, there are several causes for congratulation to be found in the mine explosion record of the past year. The 3-month period November 1, 1930, to February 1, 1931, was one of the worst 3-month periods in the history of bituminous coal-mining in the United States in point of occurrence of major disasters (there having been seven major disasters with 173 fatalities), but the succeeding 5-month period to the end of the fiscal year had unquestionably the best record of any similar period in the past 30 years, as there was not a single major disaster in any bituminous coal mine and there was but one in anthracite mines, resulting in loss of five lives. This immunity from major disasters during the late winter and spring months is unusual; these are the months in which some of the worst disasters in coal mining in the United States have occurred. At least two gas explosions were stopped by rock-dust, and there is very good cause to believe that the fatality list for the year would have been extended by at least 100 had it not been for rock-dusting.

It is a cheering fact that Colorado, Kentucky, and Utah do not appear in the list of States having explosions during the past year, and that Alabama and Washington had no fatalities from explosions. When the hundreds of gassy and dusty bituminous mines of Pennsylvania can operate for nearly 2½ years without a major explosion disaster, the person who believes that explosions of a widespread nature can not be prevented, will certainly be forced to revise his belief; unquestionably two of the main contributing causes to this very excellent record of the bituminous mines of Pennsylvania are the increased activity in the inspection of the installation and use of electricity in bituminous mines, and also the extension of the use of rock-dust. When the bituminous mines of Pennsylvania with over 130,000 employees and with hundreds of gassy mines can operate nearly 2½ years, producing much over 300,000,000 tons of coal without a major disaster, and when all of the bituminous and lignitic mines of the United States can operate more than eight months (from January 28, 1931, to the date of writing this paper) without a major disaster, it is futile for anyone to deny that widespread explosions are preventable. There is no question that such occurrences can be avoided, and there is every reason to believe that within the next few years it will be found that the death rate from explosions in the mines of the United States will be reduced at least 50 per cent and very probably as much as 90 per cent.

I. C. 6541 October, 1931

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

MILLING METHODS AND COSTS OF THE CONIAURUM MINES, (LTD.), SCHUMACHER, ONTARIO

BY

John Redington

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MILLING METHODS AND COSTS OF THE CONIAURUM MINES (LTD.), SCHUMACHER, ONTARIO[1]

By John Redington[2]

INTRODUCTION

This paper describing the milling practice at the Coniaurum Mine is one of a series of papers on milling methods, practice, and costs prepared in cooperation with an published by the United States Bureau of Mines.

The 500-ton mill of the Coniaurum Mines (Ltd.) is located at approximately the center of the company's property and a little more than a mile from the town of Schumacher, Ontario. The main shaft through which all the ore is hoisted is adjacent to the mill so that the ore hoisted in a 3-ton skip is dumped directly into a 15-ton bin at the crusher.

CHARACTER OF THE ORE

The ore as broken in the mine is comparatively fine, as the stopes are narrow, and consequently no large breaker is necessary. It consists of a mixture of quartz and mineralized schist carrying about $6 gold per ton and varies in size from 10-inch slabs down to fines. There is a very large proportion of fines which are very wet and at times give considerable trouble in the process of dry crushing. Little hope is held of improving the latter condition unless the veins should widen and thus allow a change in stoping so that the ore can be broken coarser.

CRUSHING PLANT

The flow sheet of the mill is shown in Figure 1.

The ore is drawn from the small bin into which the skip dumps by a "live-roll grizzly." This machine acts as a feeder to the gyratory crusher and at the same time removes the minus 1¼-inch material which would give trouble and reduce the capacity of the crusher. This grizzly was installed some months after the plant was started and considerable thought was given to the type of machine to be used. It was realized that a vibrating screen under proper conditions would remove a greater percentage of fines, but it was considered that with such a variety and size of feed (some slabs are 10 inches thick and 18 to 24 inches long) that the traveling grizzly would be more rugged and less costly to operate. In addition, a more uniform feed could be maintained, and our experience had shown that this was necessary for the best operation of the crusher.

The crusher is a 10-inch McCully type fine reduction gyratory and is directly connected to a 100-hp. motor. It is set for reduction to 1-3/4 inches and discharges onto a conveyor where the crushed product unites with the undersize from the grizzly. The ore is conveyed under a 36-inch lifting magnet to a centrifugal-discharge 18-inch bucket elevator. The latter raises the

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from the U. S. Bureau of Mines, Information Circular 6541."

2 - One of the consulting engineers, U. S. Bureau of Mines, and manager, Coniaurum Mines (Ltd.).

ore to a cylindrical steel bin of 700 tons capacity. A bin of such large capacity is used so that the crusher can be operated at one period and the rolls at another and thus save power, which is purchased on a peakload basis.

The ore is drawn from chutes in the bottom of this bin by conveyor belt and fed to the primary rolls. These are heavy duty 62 by 20 inch rolls fitted with 5-inch forged chrome-steel shells set to ½ inch. From the rolls the ore is conveyed to a centrifugal-discharge 18-inch bucket elevator which feeds two mechanically vibrated 4 by 5-foot screens. When the ore is very wet most of the trouble occurs at this point.

On these screens a punched plate is used with 7/16-inch round openings; the advantages of wire-cloth screen are realized, but the former type has been found to be best suited for the type of vibrator used and the character of the ore. The life of a screen is 6 weeks.

The oversize from the screen passes to a secondary roll in a closed-circuit arrangement, and the discharge of the secondary roll joins that from the primary roll. The undersize from the screen drops on to a conveyor and goes to the main 1,000-ton bin, in the cyanide plant.

Following is the screen analysis of the product as it leaves the crushing plant:

Mesh	Per cent	Mesh	Per cent
Plus 4	11.14	Plus 35	5.85
Plus 6	14.02	Plus 48	2.82
Plus 8	12.23	Plus 60	2.40
Plus 14	19.53	Plus 100	3.02
Plus 20	8.90	Plus 150	1.45
Plus 28	6.13	Plus 200	0.94
		Minus 200	11.57

The secondary rolls are the same as the primary but are run at a higher speed and are set as close as it is possible to keep them. This setting of the rolls is governed by the power taken to drive them, as shown by the ammeter. The operator controls the feed also by the ammeter and forces it at maximum rate at all times. In this way the shells are worn uniformly by the crushing action of the ore itself and require very little dressing. No cutting of shells has been necessary and those on the primary rolls show very little grooving while on the secondary rolls they have a smooth surface at all times. The little dressing of the shells that is necessary is done with carborundum bricks on the edges while running. The shells wear down until they are 3/4 to 1 inch thick when they loosen or start to spread and are then discarded.

Each roll is driven separately by a 16-inch belt from a 35-hp. motor. The motors on the primary roll are fully loaded and on the secondary rolls are about 30 per cent overloaded. The capacity of the rolls varies a great deal, and when working on a very wet ore, as mentioned previously, it is reduced considerably. The average capacity on all ore is very close to 45 tons per hour. The coarse crushing rolls operate at a peripheral speed of 1,090 feet per minute, with new shells, and the fine rolls at 1,330 feet per minute.

All conveyors use a 24-inch belt and are equipped with Timken tapered roller-bearing idlers.

CYANIDE PLANT

The bin in the cyanide plant is of the cylindrical steel flat-bottom type. The ore is drawn by conveyor from chutes in the bottom and is fed by launder with cyanide solution to a 30 by 6 foot Dorr duplex classifier in closed circuit with a 5 by 16 foot tube mill. Two grinding units are available, but at present only one is in use. The classifier is placed

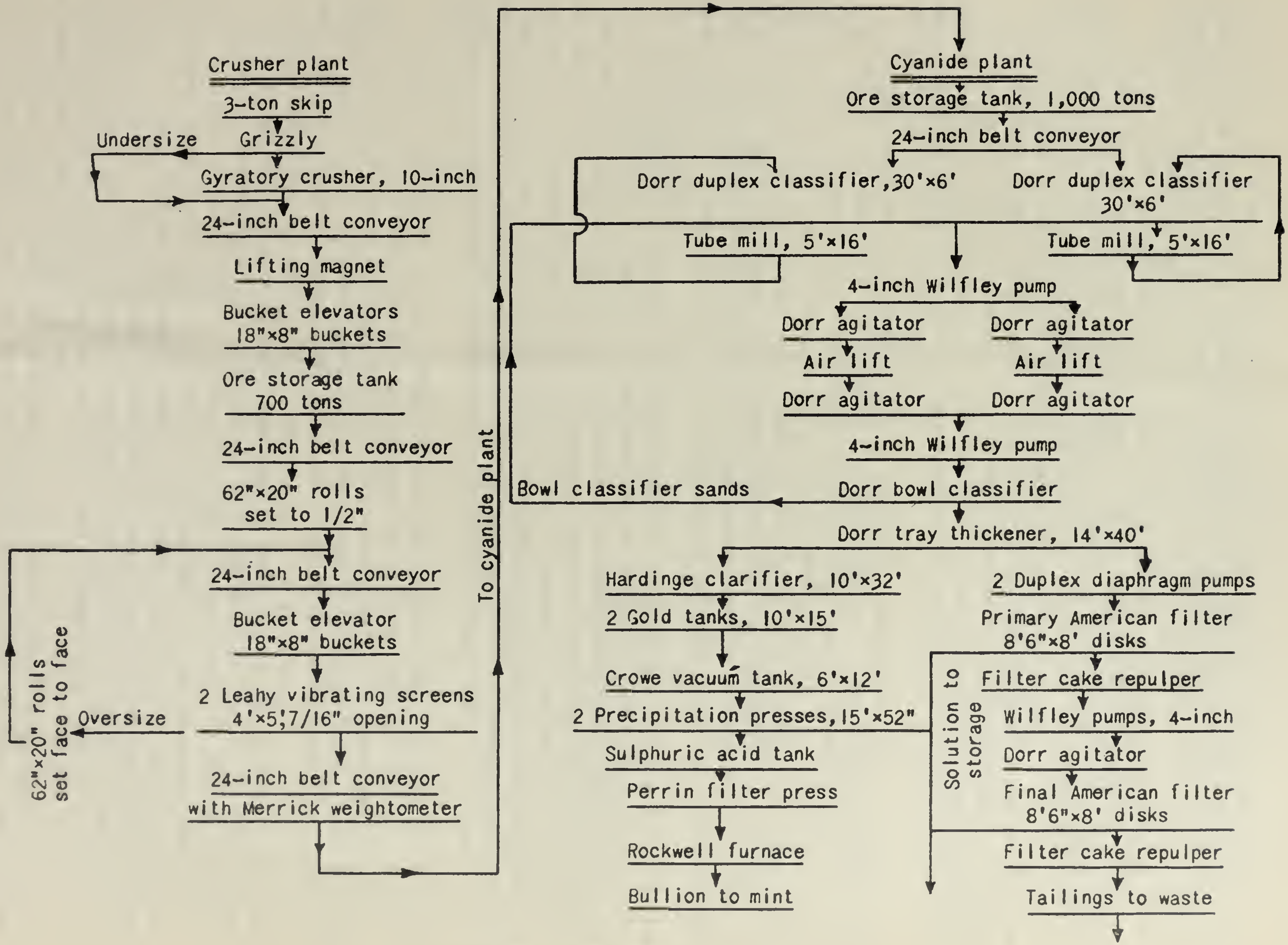

Figure 1.- Flow sheet of Coniaurum mill

at a slope of 3 inches to the foot, and the rakes are run at 24 strokes per minute. The density of the overflow is 1.2 parts of solution to 1 of solids. A screen analysis of the classifier overflow shows 10 per cent plus 65-mesh, and at this point 75 per cent of the gold recovered is already in solution.

The tube mill is operated at 28 r.p.m., with a moisture content of 28 to 30 per cent. It is lined with a chilled white-iron liner 2½ inches thick of a patented type called the "pocket liner," which is cast in this vicinity. The grinding media are 2-inch balls forged from steel rails, and the consumption is 1.8 pounds per ton of ore. The drive consists of a herringbone gear driven by pinion directly connected to a 150-hp. motor.

The working strength of the solution is kept at 1.25 pounds of KCN per ton, and alkalinity at 0.75 pound per ton in terms of CaO. The cyanide used is Aero brand and the consumption is 0.5 pound KCN per ton of ore. The cyanide is added at regular intervals in weighed quantities along with the classifier sands entering the tube mill. The lime is added dry to the ore as it is fed to the crusher.

The pulp overflow from the classifier is pumped by a 4-inch centrifugal sand pump to a 20 by 24-foot Dorr agitator. The speed of the rakes is 8 r.p.m., and the pulp is drawn from the bottom and raised by air lift to the second agitator, giving a contact of 16 hours. There are four agitators, but at present only two are being used with the one tube mill. The pulp density is the same as that overflowing the classifier, and, although it is very coarse little difficulty is experienced in keeping it in suspension. Compressed air is supplied at 25 pounds per square inch by a single-stage compressor driven by a 60-hp. motor. From the second agitator the pulp is pumped by a 4-inch centrifugal sand pump to a 17-foot diameter bowl classifier.

This classifier is supported by steel columns on a floor above the tube mills, and the sand containing about 20 per cent of moisture is run by chute back to the tube mill. The overflow is the final product from the grinding, agitating, and classifying circuit and at this point is determined the fineness of grinding. The economical point of grinding has been determined to be 60 per cent minus 200-mesh; a daily sample of the overflow is taken for sizing test, and the fineness is controlled by regulating the specific gravity of the pulp overflow, which is kept at a density between 2.1 to 1 and 2.5 to 1, depending upon the tonnage, value, and character of the ore. The reciprocating rakes run at 16 r.p.m. and the bowl rakes at 5 r.p.m.

The bowl classifier, as can be seen is in closed circuit with the agitators and tube mill, and in this position acts as a concentrator and gives an opportunity to grind selectively the sulphides with which the gold is closely associated. Its importance in the flow sheet is shown by the value of the samples taken of the different products. With the pulp coming from the agitator assaying 60 cents per ton the value of the sands from the bowl classifier going back for regrinding ranges from $3.75 to $4. a ton.

The overflow from the bowl classifier runs by launder to a 40 by 14-foot Dorr tray thickener and the thickener pulp is drawn by Diaphragm pump and fed to an 8-disk 8-foot 6-inch American continuous filter. Here it is filtered and given a barren solution wash, and the cake drops into a repulping machine. This consists of a shallow wooden tank 12 feet 6 inches by 30 inches by 28 inches fitted with a horizontal shaft on which are fastened heavy iron paddles. Barren solution flows into the tank, and as the paddles revolve the cake is broken up into a pulp and flows from the tank. The height of the pulp is kept below the bottom of the shaft, and the bearings are set outside the machine so that no trouble is experienced from slime getting into them.

The pulp, which as stated above is mixed with barren solution, is next pumped to an 18 by 20 foot Dorr agitator and drawn from this agitator to a second filter similar to the first one. On this filter it is given a water wash, and the cake drops into another repulper, is mixed with water, and runs by gravity to the tailings pond.

A two-stage Rees Roturbo pump is used with each filter unit; in fact, all solution pumps are of this make and are either single-stage or two-stage. The filtrate can be pumped to the circulating solution tank or taken into the circuit at other different points when necessary. Vacuum for the filter is furnished by a 23 by 12 inch duplex pump driven by a 75-hp. motor.

On the primary filter a row of sectors is taken off every second day and placed in a small wooden tank with a dilute solution of muriatic acid to remove the lime. On the secondary filter where water wash is used and lime formation is more rapid, a row of filter bags is treated every day. This gives a period of 20 days per filter bag on one filter and 10 days on the other. The bags are repaired by hand by the operators and the average life of each, including the repaired ones, is 130 days.

The overflow from the Dorr thickener runs to a Hardinge sand clarifier. The sand through which the solution filters is all minus 1/4-inch, with the following screen analysis:

Mesh	Per cent
Plus 20	29.05
Plus 100	67.45
Minus 100	3.50

In operation, the solution is allowed to filter through the sand bed until the rate of filtration is slower than the rate of precipitation. The scrapers are then revolved, scraping the slime to the center and drawing it from the bottom of the tank. If this does not increase the filtering capacity quickly enough, the scrapers are lowered from 1/16 to 1/8 inch, and that much sand is cut off the top of the filter bed. We have found this a very satisfactory method of clarification - one requiring little attention and low in cost. A new charge of sand which is still in use was put into the tank in March, 1929, and should last until the summer of 1931.

From the clarifier the solution is pumped by centrifugal pump to the 10 by 15 foot gold tanks, and then from the Crowe vacuum system to the precipitation press. This is similar in all respects to the customary practice except that a centrifugal pump is used in place of a reciprocating pump to deliver the pregnant solution to the press. The problem of eliminating the air from this type of pump is overcome by connecting the glands of the pump by a small pipe line to the Crowe vacuum tank through which any air that enters the pump is drawn away by vacuum.

Six hundred and fifty tons of solution is precipitated every 24 hours. The consumption of zinc dust is 0.025 pound per ton of solution, and a precipitate is obtained averaging $175 per pound. Lead nitrate is added to the clarified solution at the rate of 1½ pounds per 24 hours.

The clean-up of the precipitate takes place once each month. It is scraped roughly from the cotton sheeting which overlies the 10-ounce duck on the press, and the sheeting is burned.

The small amount of ashes with flux goes directly to the furnace. The precipitate is given a sulphuric-acid treatment, forced into a press with air, washed, and dried with air to about 35 per cent of moisture. It is then fluxed with soda ash, borax, fluorspar, and silica and refined in a Rockwell furnace. A proportion of the slag is rerun or used with the flux, and the excess slag which amounts to very little is returned to the tube mill. The resulting bullion averages 850 fine in gold and 110 fine in silver.

The recovery for the full year 1930 was 96.35 per cent; and the costs, without depreciation, for the same period are given below:

Total tons milled	122,972
	Cost per ton
Crushing, screening, and conveying to cyanide-plant bin	$0.346
Tube milling and classification	.202
Agitating and thickening	.072
Clarification and precipitation	.042
Reagents	.087
Double filtration	.113
Tailings disposal	.015
Refining	.016
Pumping	.046
Total	0.939

CONSUMPTION OF MATERIALS AND REAGENTS

Ball consumption	1.8 pounds per ton of ore milled
K.C.N. consumption	0.5 pounds per ton of ore milled
Zinc dust consumption	0.025 pounds per ton of solution (650 tons solution per 24 hours)
Lead nitrate	1.5 pounds every 24 hours
Life of vibrating screens	6 weeks
Life of filter bags	130 days

I. C. 6542

November, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING LAWS OF LATVIA

BY

E. P. Youngman

I.C. 6542
November, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE--BUREAU OF MINES

MINING LAWS OF LATVIA[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper presents one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the rights of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of Latvia has been prepared entirely from a translation of a reply to a questionnaire of the United States Bureau of Mines, made in the Latvian Ministry of Finance, forwarded by Frederick W. B. Coleman, the American Minister at Riga, and transmitted to the Bureau of Mines through the courtesy of the Department of State.

INTRODUCTION

The mining laws of Russia (Collection of Laws, vol. 7, 1912) are still in force in Latvia. The Free State of Latvia (declared independent on November 8, 1918) is composed chiefly of the Baltic Provinces of the former Russian Empire, which are ethnographically Lettish: The Province of Courland, four districts of the Province of Livonia, and three districts of the Province of Vitebsk. Since the establishment of its independence, Latvia has not felt the need of a special mining law, inasmuch as mining so far has not been found profitable in that country.

A supplement to this paper, entitled "Mineral Resources," is a brief summary of the minerals found and utilized in Latvia.

RIGHTS OF FOREIGNERS

A foreign individual is permitted to explore for minerals and to own and operate mines in Latvia on equal terms with Latvians.

A foreign corporation is required to establish a branch in Latvia, which shall be subject to the corporation laws of the country. A certain number of the partners or incorporators must be citizens of Latvia.

1 - The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6542."

2 - Rare metals and nonmetals division.

OWNERSHIP

The owner of the surface land is owner also of everything beneath it. Even on private lands, permission from the Government to exploit minerals is required.

PROSPECTING

Prospecting on private land may be done only with the consent of the landowner; the State does not impose any special terms upon either the prospector or the owner of the surface.

To prospect on State-owned land, the consent of the Government is required "if the exploration is connected with the cutting of trees, drilling, and other earth work." The following regulations apply to prospecting permits for State land.

A prospecting permit, giving exclusive right to the permittee to explore in a given area, is issued by the Ministry of Agriculture to any one willing to comply with all legal stipulations. The law does not prescribe any definite qualifications for a permit holder. A permit covers a definite area for a specified mineral or minerals: 4 square versts[3] (about $1\frac{3}{4}$ square miles) for any mineral except petroleum ("naptha"), for which 90,000 square sagenes[4] (about 0.16 square mile) are allowed.

A permit to explore for petroleum, or naptha, is valid for 1 year; a permit for any other mineral may extend over 2 or 3 years.

In order that a prospecting permit may remain in force, it is necessary that work thereunder be commenced within a year of the issuance of the permit and that no interruption of work be of more than one year's duration.

A fee (beginning with the second year) of 30 lats[5] (about $5.75) is levied for each area.

A prospecting right may be transferred to another person, provided notification of the proposed transfer is made to the Commercial and Industrial Department, which is required to make the necessary notations on the permit.

MINING

A permit to mine on private land, as well as on State land, is required. Such a permit, or concession, is issued by the Commercial and Industrial Department.

A mining right, which may be transferred only with the consent of the Commercial and Industrial Department, is granted for an unlimited period--until exploitation shall have been completed.

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3 - A square verst equals 0.4394 square mile.

4 - A square sagene equals 49 square feet.

5 - The estimated value of the lat in U. S. currency in 1929 was 19.3 cents.

An area of 1 square mile may be allotted on State lands for all minerals except petroleum ("naptha"), for which from 1 to 10 dessiatines[6] may be granted.

Forfeiture of a mining right may result if the holder thereof does not pay the rent due upon State land, does not undertake operations within the legally prescribed time, does not produce the quantity of minerals required by the Commercial and Industrial Department, or does not fulfill any other provision of the law.

TAXES AND RENTS

The explorer is required to pay, in addition to a mining tax, a rent equivalent to the average received by the State for the land in question for the preceding three years.

SUPPLEMENT

Mineral Resources

Latvia (24,440 miles square, slightly larger than the State of West Virginia) is not mining country, 70 per cent of its population being engaged in agriculture. In the latter part of 1928, the Latvian Department of Industry stated that, with the exception of peat, clay, gypsum, limestone, and similar rock strata, no mineral deposits of any importance existed in Latvia. One concession conveying the right to prospect for oil and another for the right to dig and collect amber along the sea coast of the Libau district had been granted, but mineral resources were confined almost entirely to dolomite and limestone (used in making cement), clay (used in the manufacturing of bricks), and some gypsum. Peat, the resources of which, according to a report made in 1921, amount to 16,000,000,000 cubic yards (covering 6,517 square kilometers), is used locally as fuel; but since it appears rarely on the market, the annual production thereof is difficult to estimate.

The following table of the country's mineral production for 1929 contains the latest available data, obtained from the report of the American consul at Riga in answer to the bureau's questionnaire.[7] As a result of increased tariff protection, an increase in the production of cement by the one factory operating in recent years was looked for in 1930. Of the total production of gypsum (35,000 metric tons), 26,875 tons was exported to Finland and Sweden.

6 - A dessiatine equals 2.7 acres (0.00421 square mile).

7 - Hurley, John P., American consul, Riga, Mineral Production of Latvia, 1929: May 9, 1930, Bureau of Mines foreign file 9948.

Estimated mineral production of Latvia, 1929[1]

Minerals	Quantity (metric tons)	Value (lats)
Cement	28,000	1,400,000
Clay	60,000,000	1,000,000
Gypsum	35,000	350,000
Iron ore	Nil	Nil
Peat	2,500	90,000
Stone:		
Dolomite	Negligible	
Marl (for cement)	No data	
Sandstone	Nil	
Other stone	No data	200,000

1 - Hurley, John P., American consul, Mineral Production of Latvia, 1929: May 9, 1930, Bureau of Mines foreign file 9948.

In addition, Rogers[8] mentions large quantities of ferruginous earth, used locally as iron ore in ancient times, until cheap transportation made it more profitable to use imported iron. Chalk formations occur in a few places, but usually the layers of chalk do not reach the surface of the earth. An area of brown coal of about 100 square kilometers was located. Limited quantities of ferric ochre have been found.

Reports refer to the finding of petroleum in Latvia in 1921 or 1922 and to the granting of an oil concession in 1928. But as yet no production has been recorded; moreover, the most recent report (February 12, 1931) definitely states that there is no oil drilling in Latvia. On May 12, 1931, the Latvian Parliament approved a regulation that ordered the admixture of alcohol in internal-combustion-engine fuels as a means of reducing the imports of motor fuels. The one oil refinery of the country operates chiefly upon the crude product from Soviet Russia.

According to Rogers,[9] clay, lime, gypsum, cement, and chalk industries have been well developed in Latvia. He says:

> In 1910 mineral products were manufactured in 150 enterprises, employing 12,000 workmen. The factories were located mostly in Riga--two of them manufacturing china, crockery, and terra-cotta wares and employing 3,390 workmen.
>
> The most highly developed industry was the manufacture of brick and roofing tiles, which comprised 71 enterprises, employing 4,579 workmen.

8 - Rogers, Leighton W., Mineral Wealth of Latvia and its Utilization: Trade Commissioner's Report, Aug. 31, 1922, Bureau of Mines foreign file 4956.

9 - Rogers, Leighton, W., Work cited.

The number of limekilns also was rather large.

The gypsum mines developed along the banks of the Bvina, Livland, Aa, and Windau Rivers produced gypsum not only for domestic needs but also for export. Forty-five enterprises, having 1,700 workmen, were producing and manufacturing gypsum, lime, cement, and chalk.

Furthermore, there were in 1910 as many as 14 glass works, employing 2,100 workmen. These factories were located mostly in Riga, Windau, and the district of Dvinsk.

In 1921, of the factories mentioned, there were in operation: 22 brick works, 1 cement factory, 8 tile and pottery works, 5 quarries, 6 glass works, and 3 looking-glass factories, employing in all 1,700 workmen.

A publication of 1929 (The Bank of Latvia) said: "The ceramic and stoneware industries, which were developed in previous years and then went through a critical period, were briskly active last year."

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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING PRACTICES, METHODS, AND COSTS AT ELKORO MINES, JARBIDGE, NEV.

BY

JOHN FURNESS PARK

I.C.6543.
November, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING PRACTICES, METHODS, AND COSTS AT ELKORO MINES, JARBIDGE, NEV.[1]

By John Furness Park[2]

INTRODUCTION

This paper, which describes the mining methods and practices at the Jarbidge mines of the Elkoro Mines Co., is one of a series of papers being issued by the United States Bureau of Mines on mining methods and costs in various districts throughout the United States.

LOCATION AND HISTORY

The Elkoro Mines Co.'s property is located at Jarbidge in the northeast corner of the State of Nevada (fig. 1). The camp is 9 miles south of the Idaho line; the nearest railroad point is 70 miles to the northeast at Rogerson, Idaho. A road leading to the south and connecting with Deeth, Nev., is passable during the summer months; the heavy snowfall which generally takes place early in the winter usually closes the road until the first of July.

The town of Jarbidge is situated in a narrow canyon at an elevation of 6,100 feet above sea level. The walls of the canyon rise steeply on each side of the camp, protecting it somewhat from very severe weather and making possible a system of mining by adits.

The original discoverer of the camp, D. A. Bourne, entered the region from the west on a prospecting and fishing trip in 1909. In panning the gravel along the creek bottom he found free gold which apparently had its source on the hillside to the east of the main valley. He staked a number of claims before returning to civilization, and by his reports created a small gold rush. Within a few months a tent town of about 2000 persons had been built in the canyon, and the hillsides for several miles around were located. The next 4 or 5 years proved to be a period of extensive speculation but rather meager development, and the camp gradually dwindled to a population of 400 or 500. Three or four mills had been built during the boom times, but the orebodies were mostly too small to warrant any such investment, and the stockholders in these ventures lost a great deal of money.

During 1915 and 1916 a number of claims were consolidated under one company and a vein was opened up which proved to be of commercial value. This property was acquired by the Elkoro Mines Co., by whom a mill was constructed in 1917. This company has operated continuously since that time, with an average mine production of between 100 and 150 tons of gold-silver ore per day.

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1 The Bureau of Mines will welcome reprinting of this article provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6543."

2 One of the consulting engineers, U. S. Bureau of Mines, and mine superintendent, Elkoro Mines Co.

GEOLOGY AND PHYSICAL CHARACTERISTICS

The Jarbidge mining district consists of a series of Tertiary volcanic flows which have undergone extensive alteration. They are cut by a large number of faults of varying degrees of magnitude and movement which make the orebodies exceedingly difficult to follow. The gold and silver is found in quartz veins having average dips of 60 to 80° from the horizontal. The strike is roughly southeast and northwest but is subject to local changes edue to extensive ground movement.

The veins vary greatly in length and width. The principal vein which has been developed in the district has been followed in ore for a distance of over a mile and no doubt extends in some form for several times that distance. Other veins, principally of a grade too low for mining, can be traced on the surface as far as 4 or 5 miles. The width of the main vein averages about 6 feet, ranging from narrow stringers up to 20 feet in rare instances.

The vin is enriched by small leads usually entering and crossing the main ledge at about a 60° angle. At these places it is often possible to widen the stopes slightly. Experience has shown that these branching or intersecting veins do not carry commercial values away from the main ore body.

An operating difficulty is usually experienced in mining the ore of split-off shoots extending into the hanging wall. The values may extend 3 or 4 feet along the diverging vein. In stoping out this ore, a shelf is formed in the hanging wall that requires a large amount of timbering and blocking.

Except at the crossings the walls of the veins are clearly defined. At these points some of the wall rock usually is broken to make sure that all the commercially valuable material is obtained. Fault zones roughly paralleling the vein and located 6 to 8 feet in the hanging wall occur at a few places in the mine.

> The economic metals of the deposits are gold and silver. They occur in the form of native gold, gold-silver alloy, or electrum, and the silver-bearing minerals argentite, cerargyrite, and naumannite. The primary metallization produced mainly sulphides, chiefly argentite and pyrite, but from the oxidized portions of the veins pyrite has practically disappeared.
>
> The chief gangue minerals are quartz and adularia, and a common feature is their microcrystalline to cryptocrystalline character. The quartz is the more important, as its dominance or abundance favors the occurrence of the ore minerals, especially gold.[3]

No ore of very high value is found, a period of years yielding a mill feed of slightly over $10. The values occur fairly regularly through the vein; that is, there are no pockets of ore, although here and there along the vein lean spots occur which, in stoping, are usually left as pillars.

During the early years of mining at Jarbidge the amounts of gold and silver recovered were about equal, by weight. As operation continued into veins further to the west, however, the proportion of silver has increased, until at present there is about ten times as much silver as gold in the finished bullion. These workings are still in the oxidized zone.

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3 Schrader, F. C., The Jarbidge Mining District, Nevada: U. S. Geol. Survey Bull. 741, 1923, p. 26.

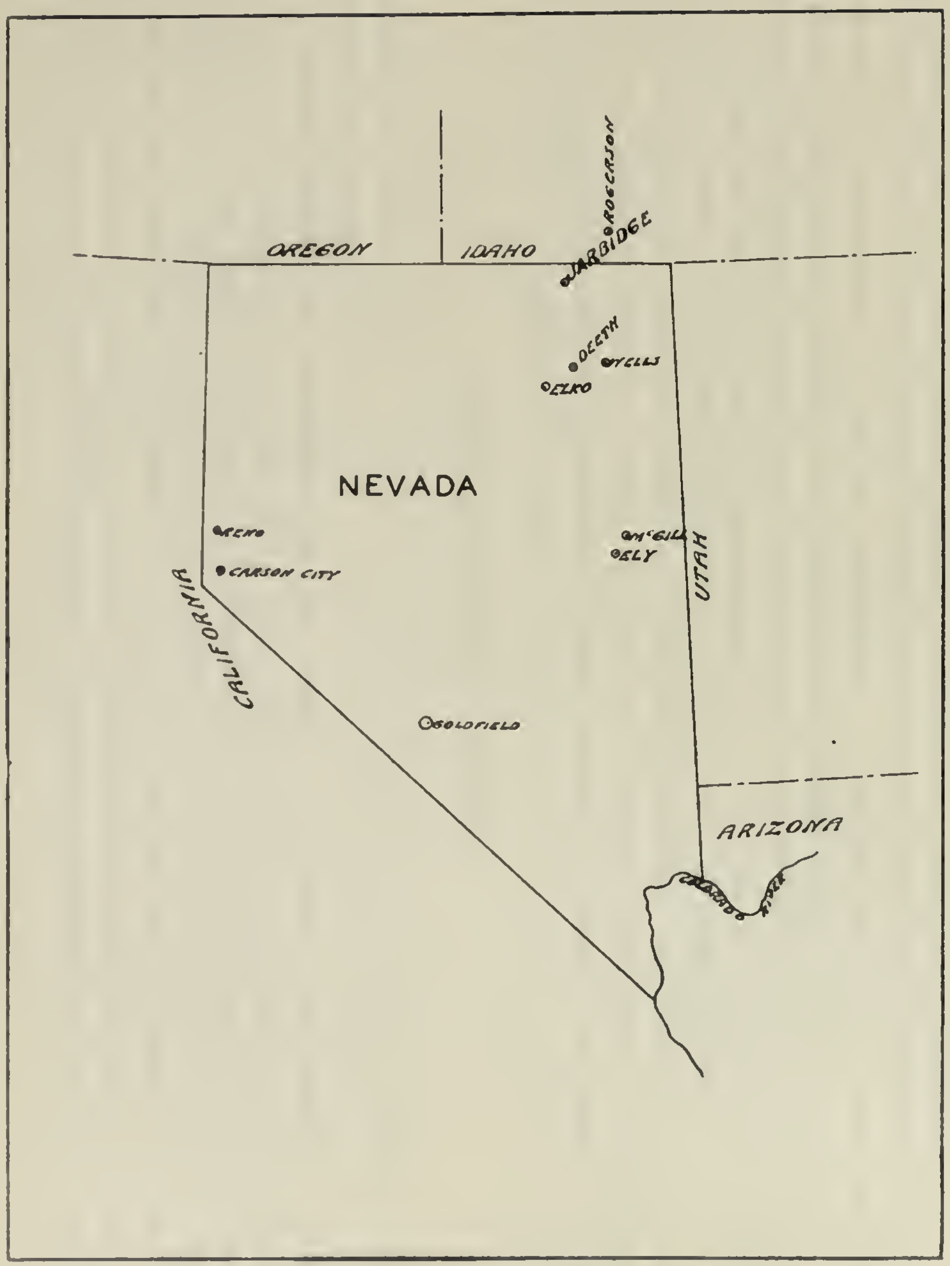

Figure 1.- Location of Jarbidge and other mining camps

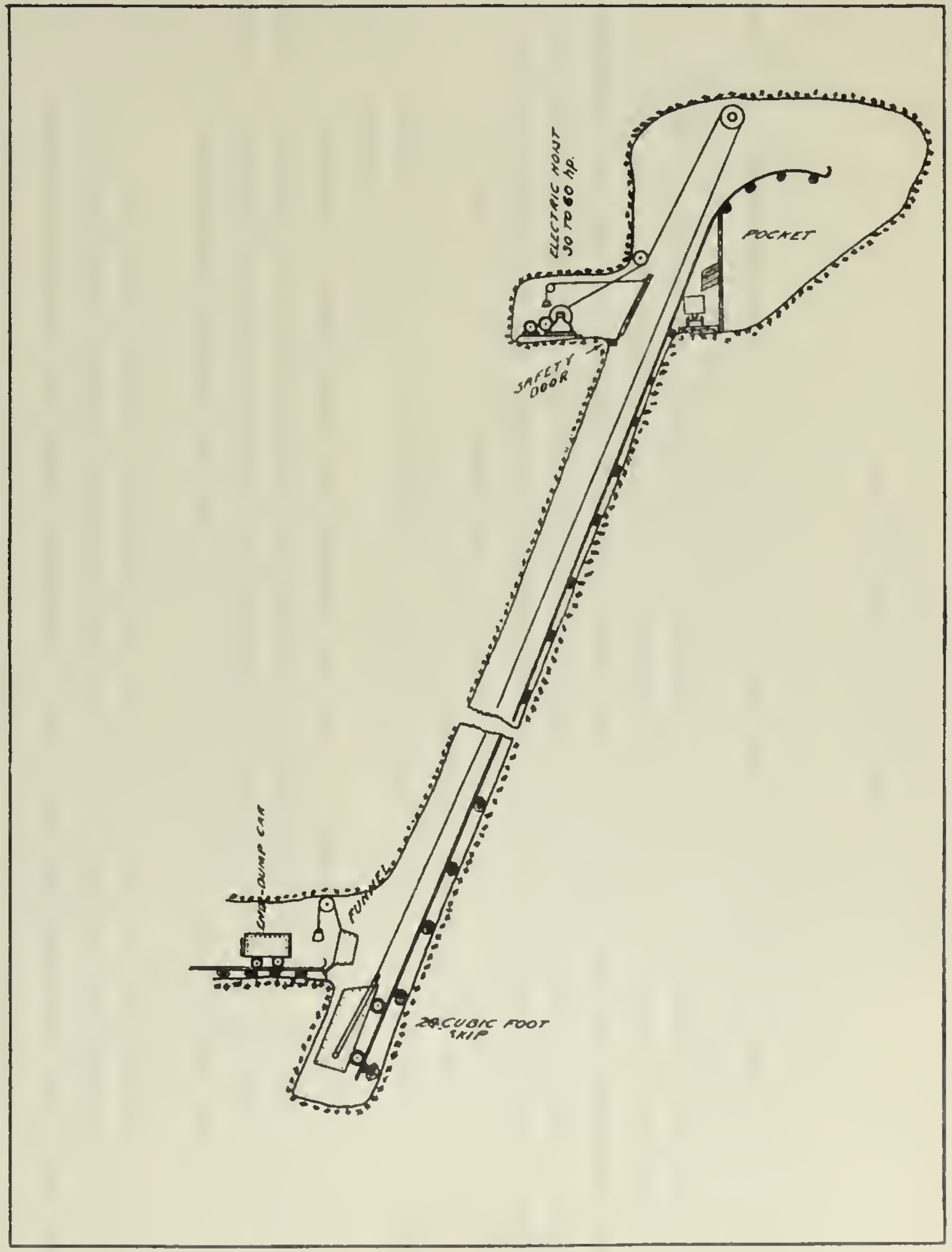

Figure 3.- General arrangement of winze

PROSPECTING AND EXPLORATION

Although the rugged nature of the country exposes numerous outcrops which are of value to the experienced geologist, the overburden, consisting principally of gravel and slides, covers the hillsides to a sufficient depth to hide the more obvious indications of vein material. During the early years of the camp, attempts were made to locate veins by a study of the values found in the surface covering. Test holes were dug at regular intervals, the dirt was panned, and the results plotted with the purpose of locating any concentration of values. The data obtained by this method proved to be unreliable, for, although gold and silver values were found in the loose overburden in the vicinity of known veins, a number of locations which showed even better pannings proved not to be associated with ledges of any kind. This condition was no doubt due to the steepness of the country as a whole, which would cause movement of the loose material for long distances without any definite trail to indicate the original source.

The Elkoro Mines Co. has taken the obvious course of having the surface geology carefully mapped in detail and a study made of all openings available over a fairly wide area. This work was done by a geologist who was thoroughly familiar with the type of formations encountered and whose conclusions have proved extremely valuable in the constant search for new orebodies.

A careful study of the rhyolite flows led to the conclusion that the veins are of value only within certain vertical limits outside of which the gold and silver content falls below the point of commercial value, although the vein itself may continue and be as strong and well defined as before. This vertical component seldom exceeds 650 feet, so that the problem of finding ore beyond any large fault zone involves not only relocating the vein itself but finding it in a flow which is favorable for the ore. This is a serious difficulty because the movement on the fault planes often exceeds a thousand feet vertically, horizontally, or both, and the rhyolite flows are very similar in appearance and characteristics.

The combination of these features tends to make prospecting and development both slow and expensive. The presence of faulting throughout the orebody, moreover, causes weak walls and spots with low mineral values, thus having a detrimental effect on ore extraction.

Diamond drilling has been done, but with only slight success. Not only are the gold and silver values irregularly distributed in the veins, but the vein material cores poorly. As a result, the numerous fault zones penetrated by the drill produce sludge which is difficult to distinguish from true vein material, and the difficulty is increased by the minor gold and silver values frequently found in such zones. Diamond drilling is consequently unsuitable as a prospecting medium. However, it might prove very useful as a means of determining the thickness, character, and relations of the various rhyolite flows in new areas.

SAMPLING AND ESTIMATING TONNAGES

Sampling is done by hand, following the ordinary practice of cutting a shallow channel across the face or back.

Development headings are sampled after each round. In stopes, cuts generally consisting of from 3 to 5 samples are made across the entire width of the stope at intervals of 5 feet throughout the length. By careful inspection of the assay results of the samples thus obtained it is possible to control the width of the opening to the best advantage.

In estimating orebodies the results of the drift samples are averaged, together with those from any raises or winzes which may pass through the ore. The widths and values thus obtained are assumed to be correct for the block. The cubical contents of the block are calculated and reduced to tons by the use of a factor which varies from 17 to 19 cubic feet per

ton of ore in place. This figure is one which has been found by experience to make the proper correction for variations in stoping widths.

The value per ton estimated for any block of ore by this method will average about 17 per cent higher than the actual extraction, but the difference is more than compensated for

by the increase in actual tonnage over that estimated. Ordinarily the total number of tons obtained from an orebody will exceed the estimate by about 20 per cent, due partly to the additional material broken as explained before, and partly to dilution caused by the sloughing of the walls during the drawing of the stopes. These factors are considered when reporting estimates, but it is almost impossible to estimate any one block of ore within 20 per cent of its final production. This is due to the continually changing width of the vein and the changes in the nature of the wall rock, which cause more or less dilution of the broken ore.

DEVELOPMENT AND MINING

No definite plan of development can be decided upon in advance, owing to the irregularity of the orebodies and the necessity of adapting the operations to local conditions. Up to the present it has been possible to open all the workings with adits from the surface; the ore is transported to the mill either by train or by aerial tramways. However, the company has recently opened up a body of ore below the level of the lowest adit and the material from this level is hoisted through a winze to a haulage aidit

The adits from the surface are spaced at approximately 150-foot vertical intervals and two of them serve as main haulage ways. The ore is either hoisted to one of these levels or run through ore passes from higher stopes (fig. 2).

All the levels are connected by raises or winzes at convenient points. These are usually driven to explore orebodies but are so placed as to serve later for stope manways or ore chutes. They also provide ventilation for stoping.

Winzes

Winzes are of the 2-compartment type and are sunk either in the vein or in the wall rock, depending on which will be more convenient for the work in view. In either case they are on a dip of approximately 70° from the horizontal which has the advantage of causing slight wear on the wheels and running gear of the skips owing to the comparatively light pressure against the walls. Each compartment is 4 by 4 feet in the clear, one side accommodating the skipway and the other the ladders, air pipe, and other equipment.

At the upper end a skip pocket is constructed which holds from 75 to 100 tons (fig. 3). It has been found advisable to make such pockets large enough so that when hoisting is done on three shifts the pockets need be drawn only on the day shift. On the lower level the ore is trammed by hand and dumped directly into the skip in the winze. Because of the short hoisting distance the skip can be pulled up, dumped, and returned in such a short time that no delay is experienced by the trammers.

To avoid excessive spilling around the sides of the skip when dumping into it, a square funnel is hinged to the edge of the track level in such a way that the car in approaching the skip knooks it into position for dumping. The trammer after dumping his car pulls a rope which raises the funnel to the upright position again where it clears the skip.

As the winzes seldom exceed 150 feet in depth the sinking buckets and ore skips are of small capacity. Ordinarily a round bucket of 1,200 pounds capacity is used during the sinking operations; this runs on a skidway of round poles of from 6 to 8-inch diameter. When the winze is to be used to hoist ore from stopes, a track is substituted for the pole skids

and the bucket is replaced by a self-dumping skip with a capacity of 24 cubic feet. It has been found necessary to build the box of the skip from extra heavy material to withstand the continual dumping from cars. This has proved to be much more damaging than loading from a chute, as there is a tendency for trammers to place large rocks on top of their loads, which naturally strike the bottom or back of the skip before there is any protecting covering of fines.

The winze hoists are of 30 to 60 horsepower, and use $\frac{3}{4}$-inch wire rope. They are placed in the hanging wall at the level of the winze collar, thus allowing the hoist operator ready access to the skip when loading supplies.

Drifts and Crosscuts

Drifts and crosscuts are driven 5 by 7 feet in the clear and are timbered with round native pine timber. Sets are placed at 5-foot intervals and lagged with either split poles or 3-inch plank cut at a local sawmill. During the past few years a shortage of available pine timber for saw logs has caused the company to resort to the use of a cottonwood found along the canyon. This wood has proved to be extremely tough and serviceable in the mine, although when green its weight makes it somewhat troublesome to handle.

The ground will generally stand for some time without timbering except in faulted sections, but in drifts or tunnels which are used for extended periods the ground tends to air-slack to such an extent that it is usually necessary to place sets and lagging throughout the greater part of their length.

Drifting is generally done on two shifts, the crew consisting of one miner and two muckers for each round. The muck is taken out by hand during the drilling of the round. It is seldom necessary to timber close to the face during the driving so that the timber crews are able to work well back of the working face and independently of the drilling crew.

No attempt is made to use standard rounds, as the condition of the rock varies greatly from one round to the next. Ordinarily from twelve to sixteen 6-foot holes are drilled, using a down cut. One-man drifting machines are used which take $1\frac{1}{8}$-inch hollow round steel The bits are formed with 1/8 inch changes in gage. All blasting in the mine is done with 40 per cent strength, low-freezing gelatin dynamite in $1\frac{1}{8}$ by 8 inch sticks and No. 8 caps. No stemming is used. The miners blast their own rounds and report to the shift boss any missed holes so that the next shift will be properly warned before going to work.

Raises

Raises are generally run in the vein and the full width of the ore, which seldom exceeds 12 to 14 feet. The length of the raise along the vein will be from 10 to 20 feet, depending on what use the raise will be put to in the future. In cases where it will serve as a stope manway the length is increased so that a chute can be built on each side of the manway. On the other hand, if the raise is put up in a low-grade section of a vein in hope of finding milling ore above, it is made as small as working conditions will allow.

Raises are timbered with round stulls spaced 5 feet apart and lagged. In order to facilitate the handling of timber and drilling supplies in the raises a narrow slide of 2-inch plank in which a small skip is operated is carried up alongside the ladder. The hoisting is done by means of small air hoists of the "tugger" type, which are easily and quickly installed at the foot of the manways.

STOPING

Two methods of stoping have been used at Jarbidge, the shrinkage and the cut-and-fill methods. The greatest part of the ore taken out has been mined by the first method and the filling of stopes has been resorted to only when the wall rock was extremely unstable. During 1930, to which year most of the costs given in this paper apply, no filled stopes were worked. It is probable, however, that at least one-third of the ore produced in 1931 will be taken from this type of stope.

Shrinkage Stoping

In starting shrinkage stopes, drifts the full width of the ore are run on the vein and a round is shot out of the back. Stulls are placed and the drift lagged over through the section to be stoped. Chutes for drawing the ore are spaced at 15-foot intervals. These are built of 3-inch native lumber and are not standardized, as it has been found that better results are obtained when the chutes are cut and fitted where they are to be used. In the building of chutes the cottonwood lumber mentioned previously has proved very serviceable because of its long-wearing qualities. Manways are placed at intervals determined by the conditions encountered or by raises which have been put up for development purposes and can be utilized for the stopes. The manways will generally average 125 feet apart, the determining factor being the distance which the hose for the stoping machines must reach to the nearest air line. In order to simplify the connecting of a number of machines at some distance from a manway, a large hose is connected to the air pipe and run across the stope where it ends in a manifold to which the individual machines may be attached. These large hose are 50 feet long and are much quicker and easier to connect than a temporary pipe.

The length of a stoping section is determined by the distance between two manways, and is, therefore, usually about 125 feet. Each section is usually carried up horizontally, but adjacent sections are not necessarily kept abreast, and their relative advance is determined by operating convenience.

Drilling is done with stopers. One-inch hollow square drill steel with a double-taper crossbit is used with all wet stopers. The vein material is drilled rather easily, as it is of only medium hardness, and it is seldom that a driller will use more than three sets of steel during a day. The work is done on the basis of a day's pay in the stopes, since the size and shape of the vein is constantly changing and it is difficult to make any definite standard for contract drilling. A bonus system based on the number of feet drilled was put in operation at one time, but it was found to be unsatisfactory, as the holes were not as carefully placed and the resulting loss in explosives more than offset any gain in drilling.

Very little plugging is done in the stopes. Such plugging as is necessary at the chutes is done by the trammers, who use stoping drills from the floor of the drift.

During the breaking of the ore about 23 per cent is drawn from the stope, which leaves the top of the broken ore at the proper working distance from the back. Very little difficulty is encountered in the form of ore arching or otherwise hanging up in the stopes while drawing. Usually when a chute fails to run, drawing an adjacent chute will bring the ore down without damage. More obstinate chutes are brought down by exploding a small amount of powder on the end of a long stick.

Shrinkage stopes are run up to as high as 250 feet, with very little timber. Pillars are left at lean points in the vein or where the wall shows signs of weakness. When the breaking has been completed (usually about 4 feet below the next level), a strong overhead bulkhead of poles is placed through the entire length of the stope to protect the timber crews who follow down the ore as it is drawn. These men place stulls across the stope to

support all loose or weak portions of the hanging wall and thoroughly clean down the walls as the ore recedes. Any pillars of valuable ore are removed if it is possible to replace them with timber. Although this method does not leave the stopes in a permanently safe condition, as the timbers rot in a matter of a few years' time, the subsequent caving is gradual and not particularly harmful to working levels.

The high timbering-labor charges shown in the cost tables are due chiefly to cost-keeping methods. All labor in stopes, except actual drilling and blasting, is done by timbermen and their helpers. Much of this, especially during the drawing off of stopes, is not timbering but mucking, yet because of the difficulty of segregating each man's time to several operations, it is all charged as timbering.

Cut-and-Fill Stopes

The filling of stopes, as explained before, is done only where conditions make it absolutely necessary. The principal occasion for filling is the occurrence of fault zones in the hanging wall, as previously mentioned, which causes too great a weight on the open stope to be supported by timber. As these conditions are well understood, filled stopes instead of shrinkage stopes are always started in such places, and in no case has it been necessary to convert a shrinkage stope into one of the cut-and-fill type.

Filled stopes are provided with cribbed ore chutes, the cribbing of sawed lumber being framed on the surface. The chutes are close enough together so that the ore can be shoveled directly into them by hand, and no cars or scrapers are used in the stopes.

Waste fill is derived from waste raises driven for this purpose at convenient points in the walls of the stope, and from development work on upper levels. The latter is dropped through the ventilation raises directly into the stope and spread by hand. As these raises are usually only one or two in number in any section, this source of fill serves only a minor part of the stope. Deflecting chutes are sometimes built under the raises to increase the spread of the ore.

In filled stopes the ore is blasted onto shoveling floors of 2-inch fir plank, which prevent excessive loss of fine ore in the waste fill.

UNDERGROUND AND SURFACE TRANSPORTATION

Track of 12-pound rails and 18-inch gage is used throughout the mine. The ore is drawn by trammers from the stopes into 20-cubic foot end-dump cars. These are trammed either by hand or in short trains by mule power and dumped into the skips, on lower levels, or through grizzlies with 12 gy 12 inch openings over the ore passes on upper levels.

On the present main haulage level ore is drawn from stopes, ore passes, or winze pockets into 26-cubic foot rocker-dump cars. Trains of 8 cars are hauled by a 2½-ton storage-battery locomotive over a distance of 1,600 feet to the mill bin.

At a level 350 feet above this main haulage level the ore is trammed to the surface in cars of a similar type and dumped into a bin. From this bin it is transported by aerial tramway to the mill.

The tramway is 1,200 feet long and uMes eight 645-pound capacity buckets. The speed ranges between 275 and 300 feet per minute, being controlled by a 15-hp. electric motor, which serves chiefly as a brake. The operating cost over a period of years has been 13.54 cents per ton. During periods when the company was operating other small mines in the vicinity, as many as four aerial tramways served as feeders for the main tramway. Two of these smaller lines were of the 2-bucket type, using 1,000-pound buckets, and had capacities of about 100 tons per 8-hour shift.

The lower principal adit is at the elevation of the townsite, so that most of the timber and supplies for the mine are taken in through this adit. An inclined tram line on the surface with a 30-hp. electric hoist is used to transport such material as is necessary to the higher levels.

COMPRESSED AIR, VENTILATION, DRAINAGE, AND SAFETY

One compressor is located at the townsite adit level and another at the principal working level for the higher workings. The compressores are connected so that either one may serve the whole mine or they may be run together. This is a decided advantage, as for short periods the air load exceeds the capacity of one machine while usually either compressor will supply all the air required. The blacksmith shop is at the adit level.

Natural ventilation is sufficient for the whole mine, since all levels are connected by raises or winzes. In development tunnels the smoke from blasting is removed by the use of a small amount of compressed air after the round is shot. In the case of long drifts or crosscuts where considerable dead air may accumulate during the driving, an electrically driven blower of the propeller type is placed at the nearest point where there is a circulation of fresh air, and a 12-inch air duct of flexible tubing is hung along the back of the tunnel level to carry the air to the face. One or two of these blowers are always kept on hand at the mine, with a supply of tubing, so that they are ready for installation at short notice.

During the last year a block of ore has been developed which lies below the level of the bottom of the canyon. It was obvious that considerable water would be encountered, and the following method has been worked out and put into operation. A winze was sunk in a particularly solid rhyolite formation in the footwall of the vein. The rhyolite not only provided strong walls in the winze itself but also tended to hold back the water during the sinking operations, so that the work was completed with the use of a single electric sinking pump in conjunction with an air sinker of fairly small capacity.

From the bottom of the winze a crosscut was run for 70 feet in a direction opposite to that which would be taken by the working crosscut connecting with the vein (fig. 4). From this point a vertical raise was put up to the level of the collar of the winze, where it connected with a crosscut which had already been driven on this upper level. A deep-well pump of a 1,000-gallon per minute capacity was installed in the vertical raise and was thus independent of the main hoisting winze. The crosscut to the ore was started at an elevation above the back of the previously mentioned water crosscut so that there was at all times a reserve water capacity equal to the water crosscut. This would probably permit a shutdown of two or three hours, which should allow time for minor repairs to the pumping equipment or the adjustment of a power failure if not too serious. An additional advantage is the fact that by the use of baffles the water in passing through the 70 feet of the crosscut can be cleared of sharp particles of grit which would otherwise greatly increase the wear on the pump runners.

The installation of this pump has been complete for only a few months so that no data are as yet available regarding its running or upkeep costs.

The mine receives its electric power over a high-tension power line from a hydroelectric plant in Idaho approximately 80 miles distant. As this line passes through an uninhabited section of sagebrush country there is always the possibility of shutdowns extending to 50 or 60 hours. This was the deciding factor in the choice of the type of pump to be used. The deep-well pump gives every indication of being entirely satisfactory, since in case of a

shutdown there is no delay in starting up when the power is restored and no harm is done to the pump itself.

These pumps operate quietly and with practically no vibration; with proper care they should give as good service as other types of centrifugal pumps.

The pump installation is so designed as to permit the addition of a second pump of equal capacity in the raise, if necessary.

No organized fire-fighting or rescue crews are maintained, as the risk of fire in the mine is small, owing to the natural dampness of the workings. The mine is kept clean and no inflammable material is allowed to collect.

A fair proportion of the men employed have received the course of first-aid treatment given by the United States Bureau of Mines, and the proximity of the mine to the company hospital permits prompt medical assistance in cases of injury.

TRUCKING SUPPLIES FROM ROGERSON TO JARBIDGE

All supplies are brought into Jarbidge by motor truck from Rogerson, Idaho. This 70-mile haul is over a dirt roadway, partly through uninhabited sagebrush country and partly following the winding canyon of the Jarbidge River. The road is occasionally closed for a few days during winter months, but no other serious difficulty has been encountered in the past three years. Because of the many sharp curves and some steep grades, trucks of rather small capacity are used.

This freighting is contracted for at a price now slightly less than 1 cent per pound. Fivo years ago the price was fixed at 2 cents, but by a gradual improvement of the road and the elimination of the worst grades at the expense of the mining company, the present rate has been made possible.

COSTS

Tables 1, 2 and 3 of costs and unit production cover in general the year 1930, with a few exceptions where figures are taken for longer periods. The tonnage shown is that trammed but the amount broken in the mine was within a few tons of the same amount.

Table 1.- Operating costs, excluding development

Mining method, Shrinkage. Tonnage trammed, 57,539.

	Labor	Supervision	Compressed-air, drills, steel	Power	Explosives	Timber	Other supplies	Total
Mining	$0.382	-	$0.048	[1]$0.114	$0.107	-	$0.208	$0.859
Timbering	.537	-	-	-	-	$0.164	.050	.751
Haulage	.295	-	-	.010	-	-	.027	.332
Hoisting	.073	-	-	.008	-	-	.003	.084
General underground	.148	$0.152	-	.007	-	-	.045	.367
Surface	.175	-	-	-	-	-	.057	.217
Total	1.610	0.152	0.048	0.139	0.107	0.164	0.390	2.610

1 Compressed air.

Table 2.- Details of mining costs on per-ton basis

Mining method, Shrinkage. Tons of ore mined, 57,539.

	Cost per ton	
Breaking (drilling and blasting)		$0.872
Labor	$0.382	
Supplies (not including explosives)	.208	
Explosives	.107	
Warehouse handling charge	.006	
Compressed air	.114	
Surface transportation	.007	
Steel and tool sharpening	.047	
Machine repairs	.001	
Timbering		.796
Labor	.537	
Supplies	.050	
Timber and lumber	.164	
Warehouse handling charge	.004	
Surface transportation	.012	
Tool sharpening	.029	
Tramming		.356
Labor	.295	
Supplies	.027	
Surface transportation	.007	
Blacksmith-shop charges	.017	
Power	.010	
Hoisting		.088
Labor	.073	
Supplies	.003	
Power	.008	
Repair charges	.004	
Sampling		.087
Labor	.024	
Assaying	.060	
Surface transportation	.003	
Ore sorting		.014
Miscellaneous labor charges		.009
Miscellaneous supplies		.005
Superintendency		.152
General		.101
Engineering		.069
Power (miscellaneous)		.007
Hospital expense		.034
Housing and feeding expense		.020
Total charges to mining		2.610

Table 3.- Operation costs in units of labor, power, and supplies[1]

Labor (man-hours per ton):	0.555
Breaking (drilling and blasting)	0.555
Timbering	.818
Haulage and hoisting	.577
Supervision	.122
Miscellaneous	.013
Sampling	.035
Total direct underground labor	2.120
General labor chargeable to underground operation:	
Ore sorting	0.023
Engineering	.053
General	.041
Average tons per man-shift	3.576
Labor, percentage of total cost	61.82
Power and supplies:	
Explosives (pounds per ton) 40 per cent gelatin	1.239
Timber (linear feet per ton)	.852
Lumber (board feet per ton)	1.355
Total power (in kw. h. per ton)	10.919
Air compression	8.477
Hoisting	1.845
Pumping	.507
Lighting	.090
Supplies and power, percentage of total cost	38.18

1 Exclusive of development.

Tables 4 and 5 of development costs show the costs per ton and per foot of the three principal types of development openings. The costs per ton are based on the tons of rock or ore actually broken and removed in the course of the development work and bear no relation to the total tonnage of ore removed to the mill. Naturally in the opening up of ore bodies The ore removed returns a profit which is credited to the development work; however, this is not taken into account here.

Drifting and crosscutting are combined as below since the two are similar in sizes and costs, and no variation is shown between development work in ore and in the country rock since practically no difference exists in the costs.

	crosscutting	Raising	Sinking
Size of excavations, feet	6 by 8	6 by 20	7 by 11
Timbered, or not	About 10 per cent	Yes	Yes
Physical properties of rock:			
(a) Hard, or soft	Fairly hard	Fairly hard	Fairly hard
(b) Firm, or loose	Firm	Firm	Firm

Table 4.- Development costs

	Drifting and crosscutting	Raising	Sinking[1]	Total development
Footage advance	5,266	616	646	6,528
Costs per foot:				
Labor	$ 6.284	$ 5.172	$15.146	$ 7.056
Supervision	.293	.293	.293	.293
Compressed air	.580	.484	.936	.605
Steel sharpening	.577	.667	.315	.531
Power	.171	.090	.434	.217
Explosives	2.094	1.071	2.106	1.999
Timber	.018	1.055	2.762	.388
Other supplies	.286	.450	.799	.353
Total underground	10.303	9.273	22.791	11.443
Outside expense	.351	.351	.351	.351
Assaying	.112	.112	.112	.112
General expense	.565	.565	.565	.565
Total cost per foot	$11.331	$10.301	$23.819	$12.471
Tons removed in development	18,431	4,620	2,665	25,716
Cost per ton	$ 3.238	$ 1.375	$ 5.773	$ 3.166

1 The figures given cover the sinking of development winzes and not permanent shafts, which naturally would show a much higher cost.

Table 5.- Development costs in units of labor, power, and supplies

	Drifting and crosscutting	Raising	Sinking[1]	Total development
Labor (man-hours per foot):				
Breaking (drilling and blasting)	2.741	2.182	5.350	2.946
Timbering	.327	3.325	3.443	.918
Shoveling	5.627	–	5.696	5.103
Haulage and hoisting	.178	2.312	7.443	1.098
Supervision	.319	.415	.310	.327
Total labor	9.192	8.234	22.242	10.392
Feet per 8-hour shift	.870	.972	.360	.770
Labor, percentage of total cost	69.08	63.34	69.71	68.77
Power and supplies (per foot):				
Explosives (pounds)	8.963	6.274	9.177	8.731
Timber (linear feet)	.440	6.002	3.065	1.224
Lumber (board feet)	–	10.063	19.576	2.931
Supplies and power, percentage of total cost	30.92	36.66	30.29	31.23

1 The figures given cover the sinking of development winzes and not permanent shafts, which naturally would show a much higher cost.

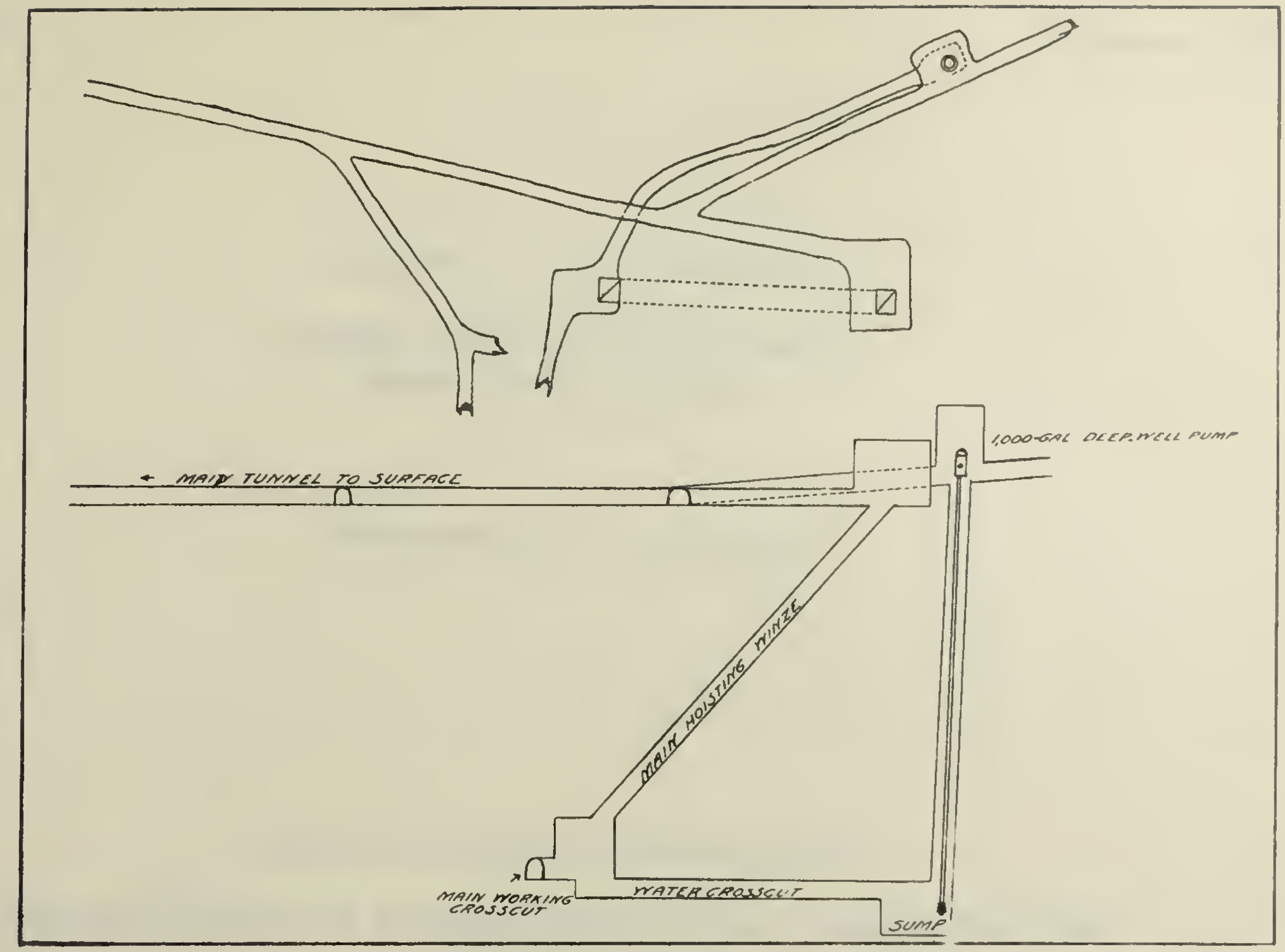

Figure 4.- Plan and elevation of pumping arrangement

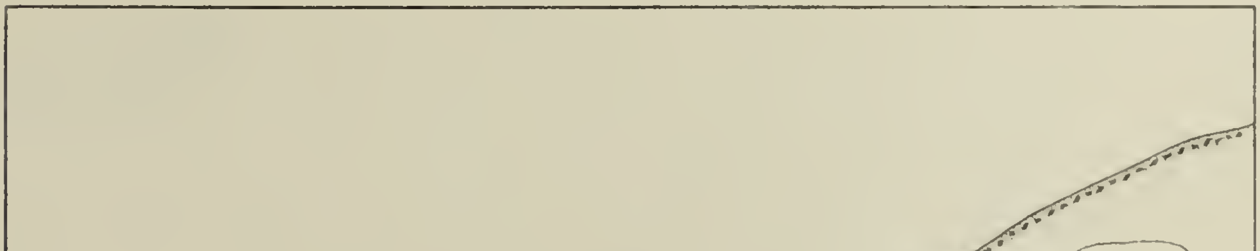

Figure 2.- Projection of direction of ore flow to mill

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Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

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MILLING METHODS AND COSTS

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MILLING METHODS AND COSTS OF THE MINAS DE MATAHAMBRE, S. A., CONCENTRATOR[1]

By A. R. Kirchner,[2] J. V. Galloway,[3] W. P. Schoder.[4]

INTRODUCTION

This paper describing the practice at the concentrator of the Minas de Matahambre, S. A., is one of a series being prepared by the Bureau of Mines on milling methods and costs.

The Minas de Matahambre, S. A., concentrator treats the production of the Matahambre mine and has a capacity of 1,300 tons of copper ore per day by selective flotation. During the year 1930 the heads averaged 4.55 per cent copper, the concentrates 29.20 per cent copper and the recovery was 95.96 per cent.

LOCATION

The concentrator is situated about 6 miles inland from the port of Santa Lucia, on the north coast of the Province of Pinar del Rio, Cuba (fig. 1).

GENERAL DESCRIPTION

The original designs were for a combined gravity and flotation plant, but as flotation methods and equipment have been gradually improved, gravity concentration has been entirely eliminated since 1928. Figure 2 shows a cross section of the concentrator.

The principal changes effected in the concentrator have kept step with improvements in flotation and milling practice, and cover the period from 1922 to the present time. As new developments have been made, they have been adopted at Matahambre whenever possible. Increase in tonnage milled and improvements in milling methods and equipment have effected a reduction in cost per ton milled from $1.508 in 1922 to $0.617 in 1930. Figure 3 is the present flow sheet of the concentrator.

Due to the various changes which have been made in operations, a comparison will be shown, in parts of this circular, between the operations of the year 1930 and work done previous to that time.

The water supply is derived from a small stream passing by the foot of the concentrator. This stream is fed by a few small springs, which very nearly run dry in the dry season, and by water pumped from the mine. These conditions necessitate a recovery of about 75 per cent of the water consumed. A description of the distribution and handling of the water supply will be found further in the circular.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6544."

2 - One of the consulting engineers, U. S. Bureau of Mines, and superintendent of concentration, Minas de Matahambre.

3 - One of the consulting engineers, U. S. Bureau of Mines, and ass't. superintendent of concentration, Minas de Matahambre.

4 - One of the consulting engineers, U. S. Bureau of Mines, and chief chemist, Minas de Matahambre.

Power is obtained from the company's power plant stationed at Santa Lucia. This is a steam-turbine generator plant furnishing two 11,000-volt transmission lines running from Santa Lucia to the substation at Matahambre.

ORE TREATED

The ore consists of primary chalcopyrite, associated with pyrite and gangue minerals of shale and quartzite. The chalcopyrite occurs as small veinlets and stringers more or less evenly distributed in the shale and quartzite or graywacke, as it is locally known, or as masses of solid mineral. The pyrite is in small stringers independent of the chalcopyrite or disseminated through it.

The quartzite, while apparently offering more resistance to crushing than does the shale, frees the mineral when crushed, while the shale, especially when wet, tends to be compressed rather than broken and does not free the mineral as easily. However, the crushing of Matahambre ore presents no especial difficulty.

The concentrator head averages 13 per cent of chalcopyrite, 7 per cent of pyrite, and 1.5 per cent of moisture.

The ore is not allowed to remain broken long enough before milling to notice any chemical changes detrimental to its floatability.

BRIEF HISTORY OF OPERATIONS

The mill was originally constructed to operate with capacity of about 600 tons daily by combined gravity and flotation methods.

The combination of gravity and flotation concentration at this time did not permit a selective treatment of all the mineral content of the ore, and concentrates produced averaged below 25 per cent.

Four Butchart primary tables and two secondary tables were operated, recovering an average of 37.5 per cent of the total concentrates, assaying 22.0 per cent of copper.

The flotation of this period was accomplished by a series of pneumatic flotation cells which proved with gradual increase in tonnage to be inadequate. These cells were later replaced by new-type pneumatic cells, and the table concentration was entirely eliminated. The use of sodium cyanide as a flotation reagent was started in January, 1929, with a marked improvement in the concentrates grade.

The change in operations caused a marked improvement in both tonnage and metallurgical results until the latter part of 1929 when a change of character of the mineral content in the heads necessitated a finer grind and a more selective float of the pyrite and chalcopyrite.

To obtain a satisfactory grind it was necessary to install one additional fine-grind mill and classifier. This additional equipment and a prudent use of flotation reagents have made possible the production of a high-grade concentrate with a recovery of approximately 96 per cent of the copper content.

The crushing plant was previously operated with two No. 6 Kennedy gyratory crushers and a No. 3 Telsmith reduction crusher, making 1-inch finished product and crushing about 600 tons daily. Then crushing rolls and screens were added, reducing the finished product to 5/8 inch and increasing the capacity to 900 to 1,000 tons per day. Later the Telsmith was replaced by a 4-foot Symons cone crusher, which resulted in reducing the finished product to 3/8 inch and increasing the capacity to 1,300 tons per day.

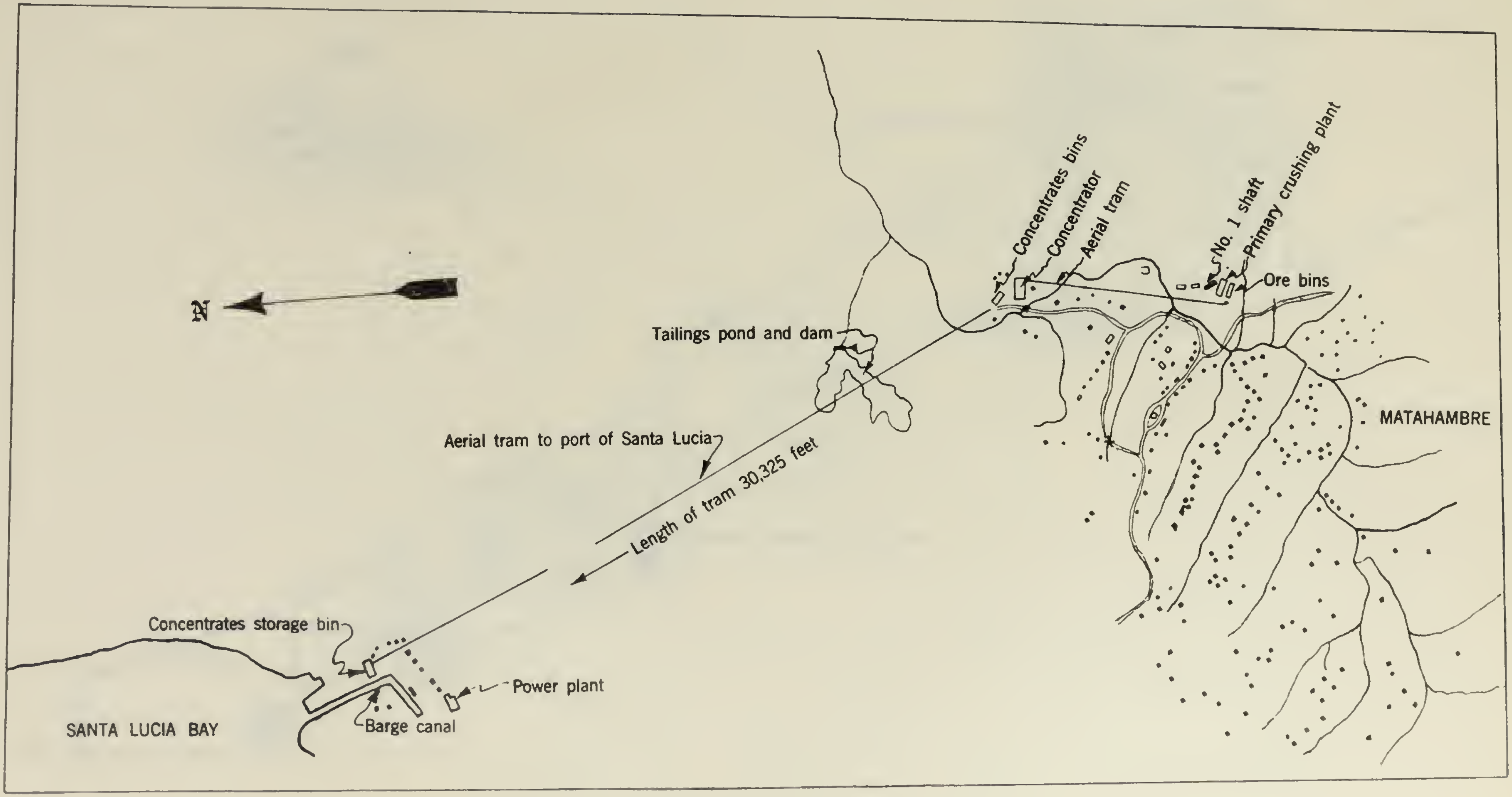

Figure 1.—Plan showing location of concentrator

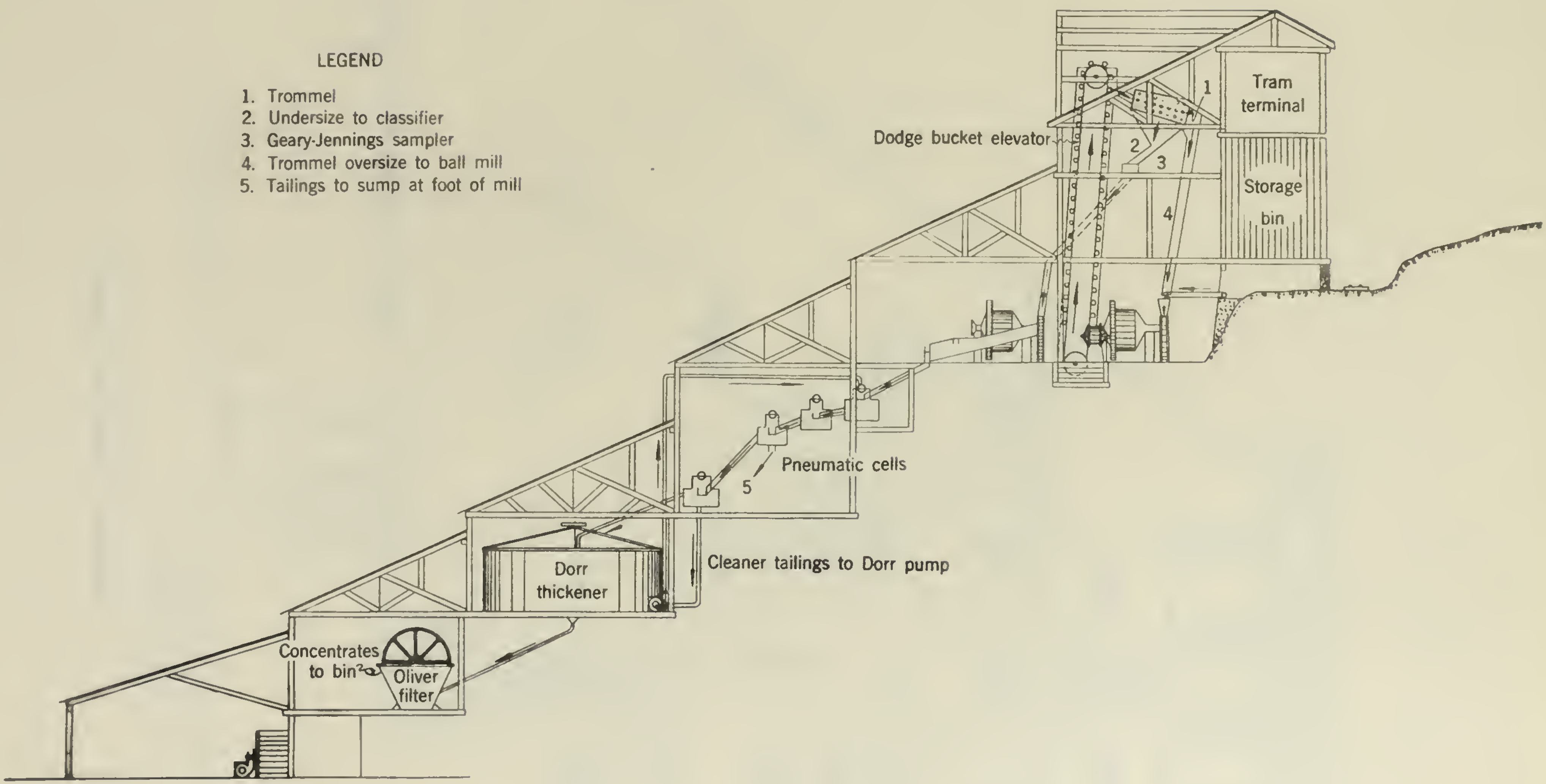

Figure 2.—Section of concentrator

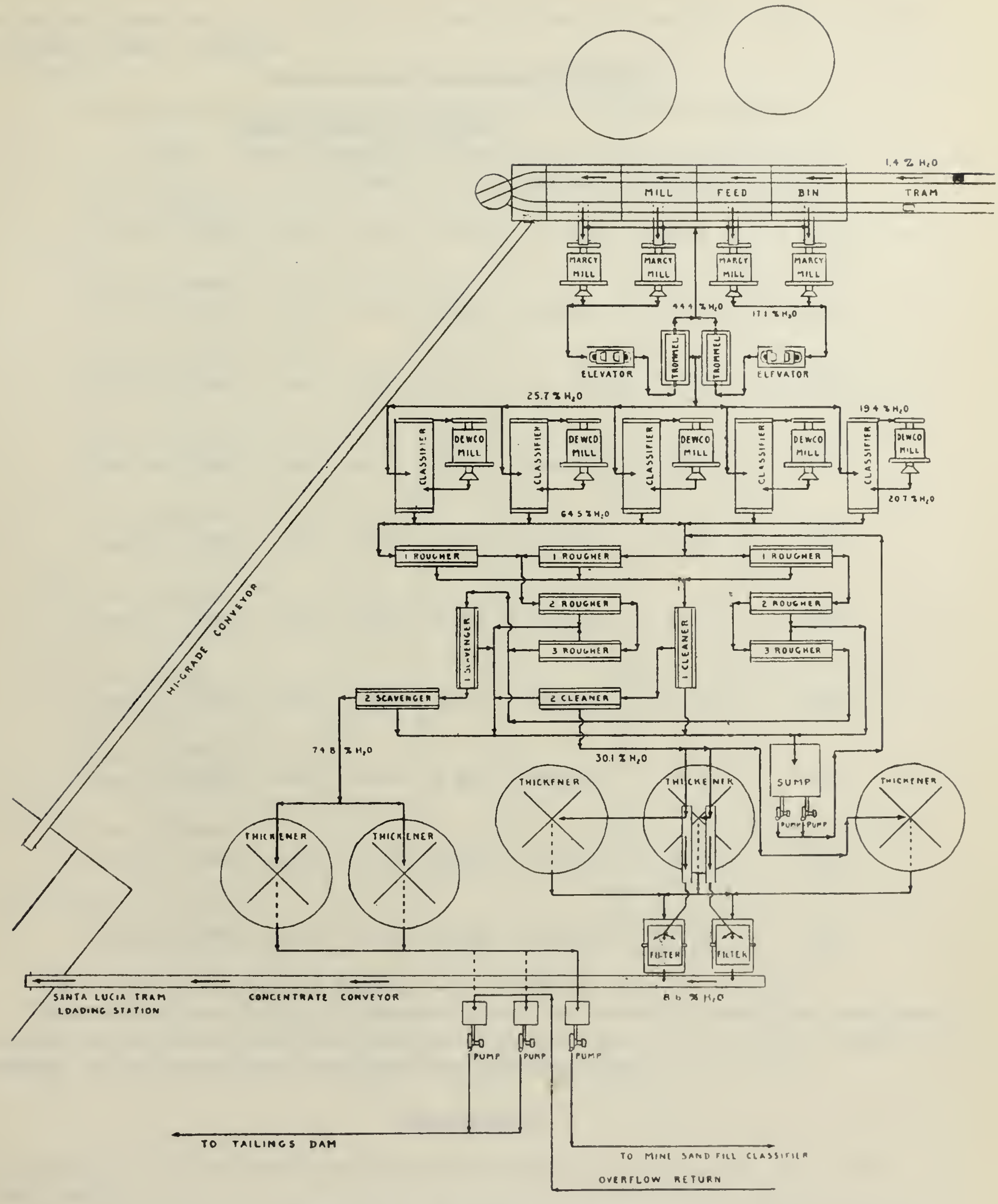

Figure 3.– Flow sheet of concentrator

The following table will give an idea of the various costs and capacity during the years from 1922 to 1930, inclusive:

Costs and capacity of plant, 1922-1930

Year	Copper content, per cent			Short tons milled	Extraction, per cent	Cost per ton
	Heads	Concen-trates	Tails			
Jan. 1, 1922-June 30, 1922	5.01	25.36	0.537	90,415	91.3	1.508
July 1, 1922-June 30, 1923	4.61	24.83	0.274	195,192	95.1	1.344
July 1, 1923-June 30, 1924	4.76	24.62	0.265	211,592	95.4	.967
July 1, 1924-June 30, 1925	4.56	24.58	0.274	224,172	95.1	.915
July 1, 1925-June 30, 1926	4.28	24.80	0.152	243,757	96.8	.955
July 1, 1926-Dec. 31, 1926	4.09	23.42	0.178	126,953	96.4	.748
Jan. 1, 1927-Dec. 31, 1927	4.60	23.73	0.271	303,098	95.2	.653
Jan. 1, 1928-Dec. 31, 1928	4.52	24.69	0.220	354,536	96.0	.571
Jan. 1, 1929-Dec. 31, 1929	4.45	27.73	0.239	356,212	95.4	.624
Jan. 1, 1930-Dec. 31, 1930	4.55	29.20	0.216	359,468	96.0	.617

The reasons for increase in milling costs in 1929 and 1930 are as follows:

Concentration was effected by combined gravity and flotation during 1928, and tables recovered 37 per cent of the total concentrates produced, with the result that reagent consumption and costs were lower than during the following years:

Reagent costs

Reagent costs per ton milled have been as follows: 1928, $.0350; 1929, .0691; and 1930, .0916.

The increased cost of reagents per ton milled in 1929-1930, however, was more than compensated for by the reduction in freight and smelting costs on the smaller tonnage of higher-grade concentrates produced.

Increased tonnage during 1929 and 1930 from the section of the mine known as the Ruisenor, during the latter two years, raised the pyrite content of the mill heads very appreciably and necessitated a much more selective float than formerly.

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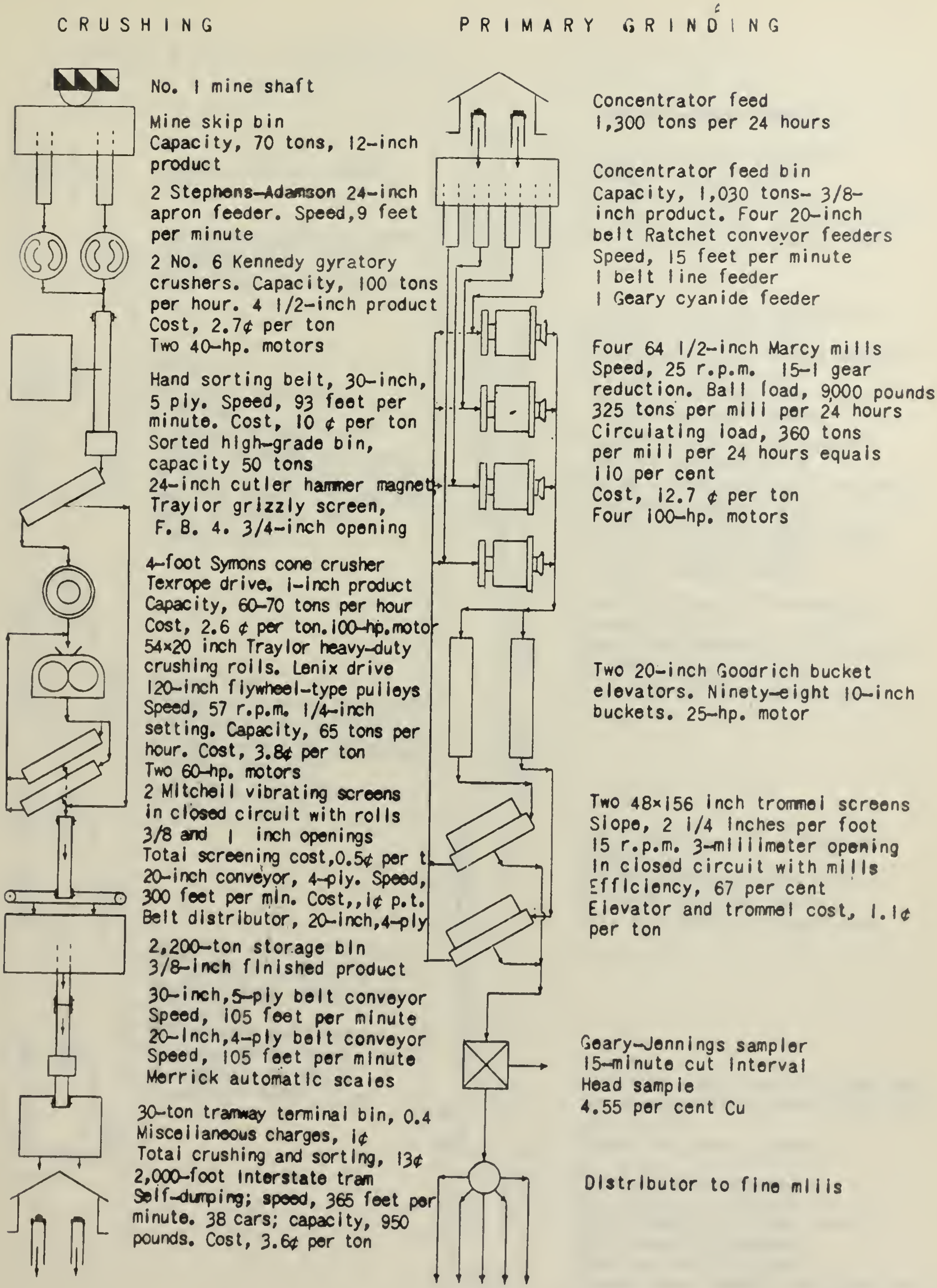

Figure 4.- Flow sheet of crushing and of primary grinding

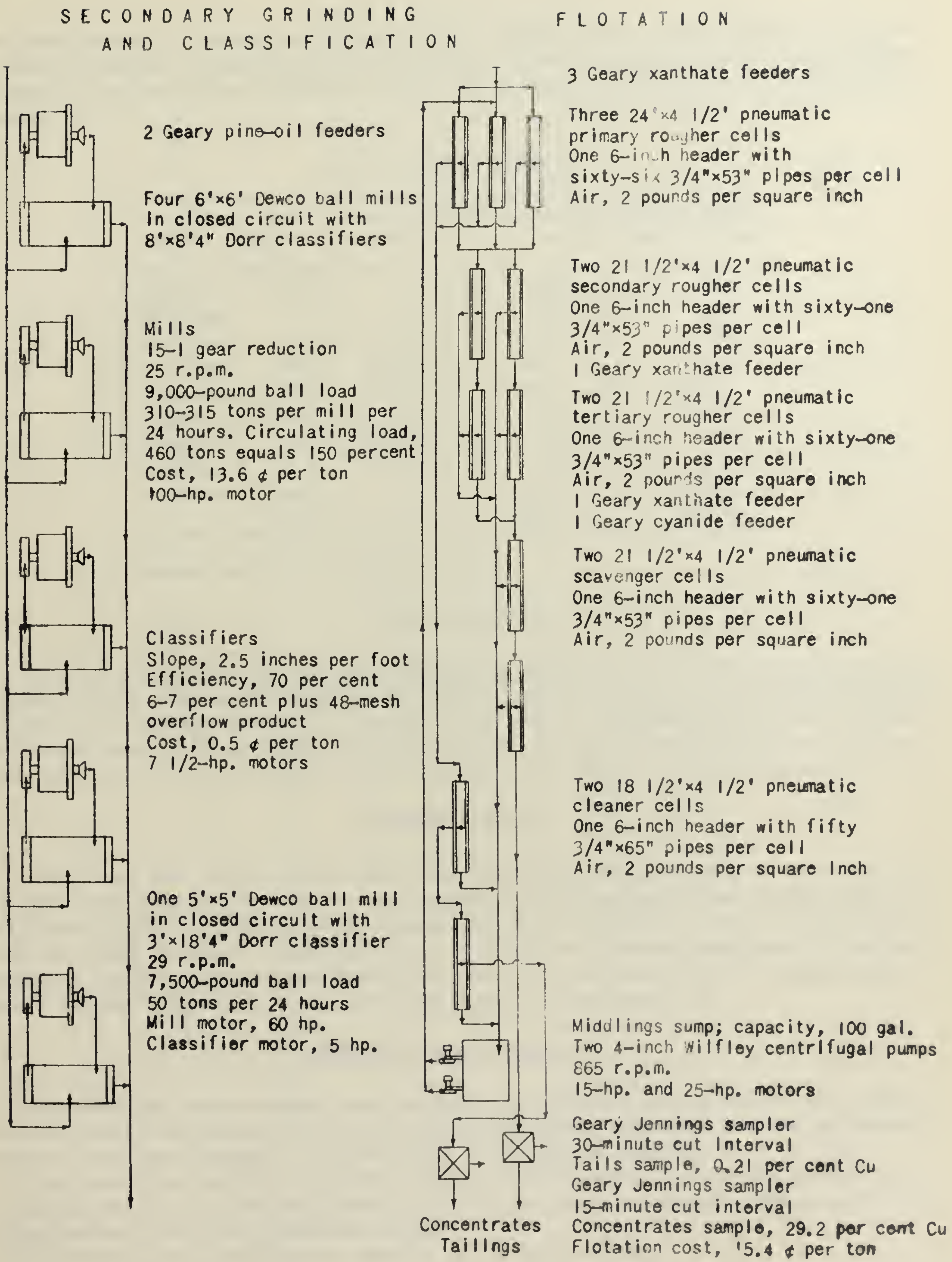

Figure 5.- Flow sheet of secondary grinding and classification and of flotation

roll shells are of chrome-molybdenum forged steel. The rolls are operated with a choke feed at a ¼-inch setting and have a capacity of 60 to 70 tons per hour of material which passes through the screen opening of 3/8 by 1 inch. The circulating load amounts to 30 tons perton hour.

ROLL AND SCREEN PRODUCTS

Size	Roll product, per cent	Cumulative per cent	Screen oversize, per cent	Screen undersize, per cent
+ 1 inch	0.0	-	0.0	0.0
+ 3/4 inch	1.5	1.5	2.2	0.0
+ 1/2 inch	9.1	10.6	15.5	1.5
+ 3/8 inch	29.0	39.6	44.5	7.4
+ 1/4 inch	23.9	63.5	22.4	20.1
+ 6 mesh	16.3	79.8	6.5	30.2
+ 8 mesh	3.6	83.4	0.5	7.9
+ 10 mesh	2.4	85.8	0.4	6.3
- 10 mesh	14.2	-	8.0	26.6
	100.0	-	100.0	100.0

Crushing plant cost, 13 cents per ton.

PRIMARY GRINDING

Primary grinding is accomplished by 64-1/2-inch Marcy mills in closed circuit with two trommel screens handling 325 tons per mill daily. The ball mill load is 9,000 pounds of 4-1/2-inch U. S. Steel Products Co. "Lorain" steel balls and manganese-steel breast liners with a ball consumption of 0.692 pound per ton and liner consumption of 0.54 pound per ton. The cost is 12.7 cents per ton. The mills are driven by 100-hp. induction motors (fig. 4).

SECONDARY GRINDING

One 5 by 5 foot and four 6 by 6 foot Dewco ball mills, each operating in closed circuit with Dorr classifiers, are used for fine grinding. The larger mills handle about 300 tons daily with a load of 9,000 pounds of 3-inch balls and a circulating load of 150 per cent. They are driven by 100-hp. induction motors. The small mill handles only about 50 tons per 23 hours. Secondary grinding costs 13.6 cents per ton. The size of feed is minus 3 millimeters and the discharge is 35 per cent plus 48-mesh material (fig. 5).

SCREENING AND CLASSIFICATION

The discharge from the coarse grinding mills is received by two 48 by 156 inch trommel screens having a 2-1/4-inch slope per foot with 3-millimeter openings. They are belt-driven with a 4 to 1 gear reduction. The efficiency is 67 per cent (fig. 5).

The classifiers are all in closed circuit with the fine mills and attain an efficiency of 70 per cent with an overflow product of 6 to 7 per cent plus 48 mesh. The slope is 2.5 inches per foot, with 27 strokes per minute.

Screening and classifying cost amounts to 1.6 cents per ton.

SCREEN ANALYSIS AND ASSAYS OF SIZES

Flotation	Tyler screen series	% of weight, cum.	Copper: Assay, cum.	Copper: Location, cum. %	Pyrite: Assay, cum.	Pyrite: Location, cum. %	Chalcopyrite: Assay, cum.	Chalcopyrite: Location, cum. %	Iron assay, cum.	Sulphur assay, cum.	Insoluble assay, cum.
1929											
Heads	+28	1.0	0.70	0.1	0.8	0.1			10.5	1.1	79.3
	+35	7.0	.76	1.2	0.9	1.1			10.7	1.3	77.9
	+48	16.3	.90	3.3	1.1	3.3			11.0	1.5	76.1
	+65	33.0	1.73	12.9	1.9	11.8			12.6	2.7	70.2
	+100	43.7	2.33	23.0	2.6	21.2			13.2	3.9	66.7
	+200	54.7	3.07	38.0	3.7	36.9			14.4	5.9	64.0
	-200	45.3	6.05	62.0	7.4	63.1			18.2	9.9	48.7
		100.0	4.45	100.0	5.3	100.0			16.3	7.3	56.2
Concentrates	+48	0.7	22.2	0.5	30.4	1.6			33.7	37.5	3.6
	+65	8.4	24.4	7.2	23.8	15.1			32.7	36.4	3.5
	100	16.4	25.1	15.5	21.8	27.1			32.3	36.1	3.3
	+200	32.4	27.0	31.6	16.8	41.5			31.6	35.3	3.0
	-200	67.6	28.9	68.4	11.4	58.5			30.8	34.3	3.0
		100.0	27.73	100.0	13.2	100.0			31.3	34.5	3.0
Tails	+20	0.2	0.42	0.4	3.4	0.1			12.3	2.3	78.2
	+28	1.3	0.42	3.0	3.4	1.7			12.3	2.3	78.3
	+35	7.3	0.42	15.5	3.4	7.9			12.3	2.3	78.5
	+48	17.6	0.35	31.6	3.7	15.8			12.4	2.4	77.0
	+65	32.3	0.29	49.4	4.4	28.8			13.2	2.7	73.5
	+100	39.0	0.27	56.5	4.7	34.8			13.3	2.8	70.2
	+200	50.7	0.25	66.3	5.0	49.0			13.5	3.0	65.9
	-200	49.3	0.13	33.7	5.2	51.0			14.1	3.1	59.0
		100.0	0.239	100.0	5.1	100.0			13.8	3.1	62.0
1930											
Heads	+35	1.0	0.14	0.03	2.05	0.3	0.41	0.03	6.9	1.23	85.8
	+48	6.0	0.25	0.33	4.00	3.5	0.72	0.33	7.8	2.38	76.8
	+65	14.2	0.58	1.82	7.22	15.0	1.68	1.82	8 4	4.43	73.3
	+100	25.3	1.53	8.51	8.37	31.0	4.42	8.51	9.1	6.00	68.3
	+150	36.3	2.46	19.57	8.27	43.9	7.12	19.57	10.6	6.89	64.4
	+200	45.5	3.20	31.93	8.15	54.4	9.25	31.93	11.6	7.58	61.9
	-200	54.5	5.68	68.07	5.70	45.6	16.42	68.07	16.4	8.76	46.2
		100.0	4.55	100.00	6.82	100.0	13.15	100.00	14.8	8.24	53.4
Concentrates	+65	1.0	25.1	0.86	6.90	0.75	72.5	0.86	25.1	29.0	12.6
	+100	8.0	27.5	7.52	7.75	6.75	79.5	7.52	28.5	31.8	5.3
	+150	19.0	28.0	18.20	10.95	22.60	80.8	18.20	30.1	34.1	3.8
	+200	30.7	28.3	29.76	12.15	40.50	81.8	29.76	31.0	35.1	3.1
	-200	69.3	29.6	70.24	7.90	59.50	85.6	70.24	29.7	34.1	2.8
		100.0	29.2	100.00	9.20	100.00	84.4	100.00	30.1	34.4	2.9
Tails	+35	1.0	0.15	0.7	2.00	0.31	0.43	0.7	6.7	1.22	72.5
	+48	6.2	0.20	5.9	2.08	2.00	0.58	5.9	7.9	1.31	71.5
	+65	18.1	0.29	24.9	3.11	8.77	0.84	24.9	8.9	1.95	69.6
	+100	32.3	0.29	44.8	4.45	22.33	0.84	44.8	9.8	2.66	69.0
	+150	43.3	0.27	55.2	5.35	36.10	0.78	55.2	10.4	3.12	66.8
	+200	51.8	0.25	61.5	5.94	47.90	0.72	61.5	10.7	3.42	66.4
	-200	48.2	0.18	38.5	6.95	52.10	0.52	38.5	13.6	3.89	58.2
		100.0	0.21	100.0	6.43	100.00	0.61	100.0	12.1	3.64	62.5

AVERAGE ACCUMULATIVE SCREEN SIZES

Tyler screen series	Crushing plant product	Marcy mills discharge	Trommel Over-size	Trommel Under-size	Open circuit classifier Overflow	Open circuit classifier Sands	Dewco mills discharge	Closed-circuit classifier Overflow	Closed-circuit classifier Sands	Dewco mills discharge	Flotation Heads	Flotation Conc'ts.	Flotation Tails
1929													
+½"	1.5												
+⅜"	8.9	0.1	0.3										
+¼"	29 0	1.0	2.1										
+3	38.3	4.5	8.2										
+4	49.0	10.8	16.0										
+6	59.2	15.6	29.5	0.0		0.0		Range					
+8	67.1	22.1	44.4	0.6		1.2	0.0	From To	0.0				
+10	73.4	31.6	65.1	7.6	Range	10.3	1.1	0.0 - 0.0	1.9				
+14	79.2	38.6	72.2	18.2	From To	24.4	6.1	0.0 - 0.1	9.2		0.0		0.0
+20	81.6	46.3	77.6	29.6	0.0 - 0.2	36.6	10.3	0.1 - 0.8	15.4		0.1		0.2
+28	85.5	54.0	81.0	39.0	0.2 - 1.2	41.2	16.2	0.5 - 3.1	21.6		1.1	0.0	1.3
+35	89.9	59.5	84.2	47.2	1.0 - 6.0	53.3	21.4	3.1 -12.2	38.8		7.0	0.1	7.3
+48	93.8	64.5	87.5	54.8	5.0 -11.5	58.0	27.2	9.0 -26.5	53.1		16.3	1.5	17.6
+65	94.4	70.3	89.6	63.4	10.0 -23.0	88.6	59.0	18.4 -40.0	83.9		33.0	9.0	34.1
+100	95.1	73.7	90.2	68.4	23.3 -38.4	90.9	65.3	29.6 -49.2	86.9		43.7	17.4	45.0
+150	96.2	76.3	91.7	71.7	35 1 -49.2	91.4	71.8	37.7 -57.1	89.0		49.1	23.5	55.1
+200	97.0	78.3	95.0	74.7	44.0 -60.0	93.5	79.8	49.0 -64.0	92.4		54.7	34.4	56.0
-200	3.0	21.7	5.0	25.3	56.0 -40.0	6.5	20.2	51.0 -36.0	7.6		45.3	65.6	44.0
	100.0	100.0	100.0	100.0	100.0 100.0	100.0	100.0	100.0 100.0	100.0		100.0	100.0	100.0
H_2O, %	1.5	18.3	17.7	45.0	74.0 -83.0	21.5	22.5	72.0 -82.0	22.5		78.5	30.0	74.0
Solids,%	98.5	81.7	82.3	55.0	26.0 -17.0	78.5	77.5	28.0 -18.0	77.5		21.5	70.0	26.0
1930													
+½"	1.5												
+⅜"	9.1		0.1										
+¼"	28.5	0.9	1.8										
+3	39.0	3.6	4.3										
+4	47.5	8.8	8.8	1.0									
+6	58.7	14.3	15.0	2.0									
+8	66.9	20.9	27.3	3.6					0.4	0.2			
+10	73.6	29.6	44.7	7.3					1.0	0.6			
+14	80.0	37.6	56.1	13.8					3.0	1.4			
+20	81.6	46.3	64.4	23.4				Range	9.3	3.4			
+28	84.4	53.7	70.2	33.3				From To	22.5	9.4			
+35	88.9	59.1	74.1	40.8				0.6 - 1.0	39.5	19.3	1.0		1.0
+48	93.0	64.2	77.7	49.0				6.1 - 7.1	62.3	35.8	6.0		6.2
+65	94.3	68.4	80.3	55.3				16.7 -18.3	74.8	48.8	14.2	1.0	18.1
+100	95.3	72.6	82.7	61.9				32.0 -33.7	83.3	60.8	25.3	8.0	32.3
+150	96.3	75.9	84.3	66.7				43.6 -46.2	87.2	68.5	36.3	19.0	43.3
+200	97.1	78.5	85.8	70.8				54.6 -58.2	89.2	75.0	45.5	30.7	51.8
-200	2.9	21.5	14.2	29.2				45.4 -41.8	10.8	25.0	54.5	69.3	48.2
	100.0	100.0	100.0	100.0				100.0 100.0	100.0	100.0	100.0	100.0	100.0
Solids,%	93.6	82.9	55.6	74.3				34.5 -36.5	80.6	79.3	35.8	69.9	25.2
SP.GR.	2.7	2.1	1.6	1.9				1.3 - 1.3	2.0	2.0	1.3	1.8	1.2
Ratio H_2O/Solid	0.1	0.2	0.8	0.3				1.8 - 1.7	0.2	0.3	1.8	0.4	3.0
H_2O, %	1.4	17.1	44.4	25.7				65.5 -63.5	19.4	20.7	64.2	30.1	74.8

HAND SORTING

Hand sorting of high-grade ore was formerly carried on intensively and was of considerable value during the period when the capacity of the concentrator was not adequate to handle mine production. At present hand sorting of high-grade ore has been eliminated, and with the exception of a small percentage of waste, which is sorted out on the belt, the mine ore is now sent direct to the crushing plant as it is delivered from the skips.

FLOTATION

The flotation circuit (fig. 5) is composed of -

3 matless pneumatic primary rougher cells
2 do. secondary rougher cells
2 do. tertiary rougher cells
2 do. scavenger cells
2 do. cleaner cells

GRADE OF CONCENTRATES AND TAILS ON FLOTATION BASIS

Cell	Air pressure	Copper in concentrates, per cent	Copper in tails, per cent
#1 rougher	1.6	28.4	0.32
2 do.	1.5	19.7	4.10
3 do.	1.7	19.7	4.80
4 do.	2.0	15.2	1.05
5 do.	2.1	14.3	1.40
6 do.	2.2	9.0	0.30
7 do.	2.0	9.2	0.27
1 scavenger	2.0	5.2	0.24
1 cleaner	2.0	24.1	17.85
2 cleaner	2.4	29.2	21.00

Total air consumption = 9,766 cubic feet per minute.
Total area of cells = 906 square feet.
Total linear feet of cells = 226½ feet.
Total air consumption per linear foot of cell = 43.1 cu. ft.

The flotation cells are supplied by a 20 by 60 inch Connersville Boston-type blower having a net displacement of 10,000 cubic feet of free air per minute at 327 r.p.m. and a maximum pressure of 2-1/2 pounds.

REAGENT CONSUMPTION

Potassium ethyl xanthate - 0.409 pound per ton.
Pine oil - 0.272 do.
Sodium cyanide - 0.031 do.
Lime (90-94% CaO) - 0.206 do.

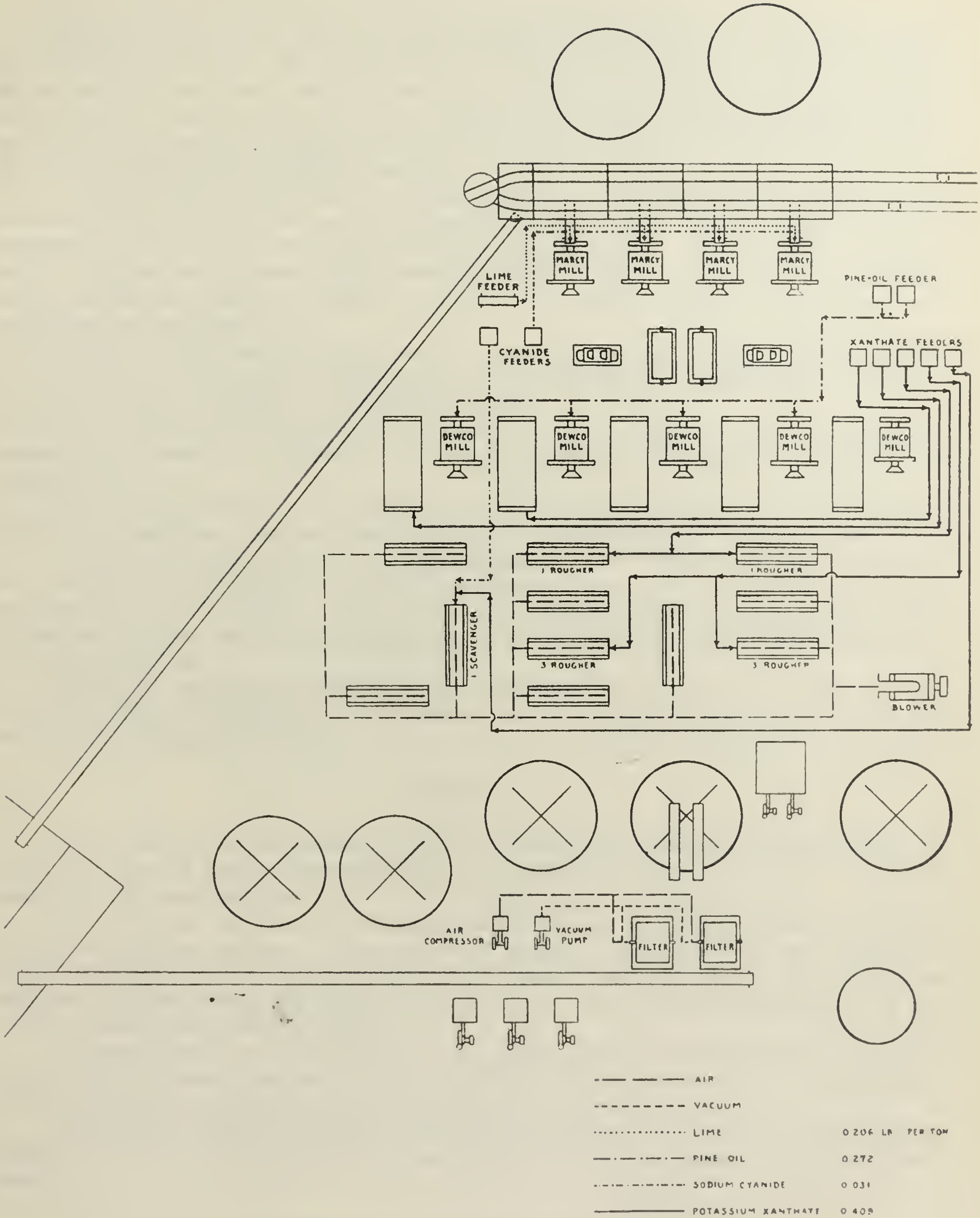

Figure 6.– Flow sheet of reagents and air

Lime is fed by a ratchet-type belt feeder to the Marcy mills.

Xanthate, pine oil, and cyanide are fed by Geary cast-iron stainless steel reagent feeder driven by 1/6-hp. motors. The pine oil goes to the fine mills while the cyanide is fed to the coarse mills and a small amount to the final rougher cells. Xanthate is distributed to various cells. Sixty per cent of the total xanthate is fed to the head of the primary roughers. The reagent cost is 9.5 cents per ton; the total flotation cost is 15.4 cents per ton. (Fig. 6 is a flow sheet of reagents and air.)

It is of interest to note that following the practice of other copper producers in the Southwest, the use of lime was adopted to depress pyrite in 1925. Numerous experiments were conducted, using line in varying amounts from 5 pound to as high as 8 pounds per ton of ore milled. Its use was continued through 1928, with the best results seeming to be obtained by using approximately 1 pound per ton of ore. Fair results were obtained, but they were never as good as had been hoped for.

In January, 1929, some tests were conducted, using sodium cyanide as a pyrite depressant with very encouraging results, and in February, 1929, its use was adopted in the mill work with a very marked improvement in concentrates grade.

After several months of operation, during which sodium cyanide was used in varying amounts and introduced into flotation at different points, the following conclusions were arrived at:

1.- That better results were obtained when using sodium cyanide in an alkaline circuit, using just enough lime to get an alkaline reaction, or approximately 0.2 lb. per ton.

2.- That sodium cyanide had to be used sparingly. With comparatively small amounts a very clean separation is obtained, but as the amount is increased the chalcopyrite losses also increase in the tailings. The best results were obtained when using approximately 0.03 lb. per ton, of which amount about 95 per cent is fed to the primary mills and 5 per cent to the scavenger cells.

3.- That the use of sodium cyanide is more economical and less messy than lime, and does not tend to blind filter canvas. The combined lime and cyanide costs are lower than the cost of lime alone was formerly.

CONVEYING

Ore is conveyed from the crushing plant to the mill by a 2,000-foot Interstate-type tram carrying 32 cars of the self-dumping type, with a capacity of 950 pounds each. The cost amounted to 3.6 cents per ton.

From the bin at the head of the concentrator, the ore is fed to the coarse mills by 20-inch belt ratchet conveyors. The Marcy mill discharge is elevated by two 20-inch belt bucket elevators to the trommel screens. The elevators have ninety-eight 10-inch buckets staggered, and are driven by a 25-hp. induction motor, which also furnishes power for the trommels, lime feeder, and distributors.

The trommel undersize is distributed to the classifiers in closed circuit with the fine mills, and the oversize returns through a distributor to the coarse mills. The classifier overflow passes directly by launders to the flotation circuit.

The middlings from the flotation circuit are collected in a sump, from which they are pumped back to the head by means of two 4-inch Wilfley centrifugal pumps driven by two 25-hp. motors.

DEWATERING AND HANDLING OF CONCENTRATES

The flotation concentrates continue to two 10-inch belt drag classifiors, tho oversize of which is spread over the tops of the filters, and the undersize dropped to three Dorr

30 by 10 foot thickener tanks. The spigot discharge passes directly to the filters (fig. 7).

The Oliver 12 by 8 foot continuous filters make one revolution per seven minutes and dewater 3.3 pounds of concentrates per 24 filter hours per square inch of filter surface. Canvas filter cloths are used, having a life of 1,200 filter hours. Vacuum is supplied by a 14 by 8 foot dry duplex vacuum pump.

The filtered concentrates are conveyed by a 20 inch by 215 foot conveyor belt to the loading bin and thence by an aerial tram to the port of Santa Lucia where it is loaded on steamers and shipped to the plant of the United States Metals Refining Co. at Carteret, N. J., for smelting and refining.

DEWATERING AND DISPOSAL OF TAILING

The tailing from the flotation circuit is received by two Dorr 30 by 10 foot thickener tanks (fig. 7).

The spigot discharge is pumped by a 6-inch Wilfley centrifugal sand pump to a Dorr type D. S. D. B. bowl classifier located above the mine workings.

The overflow from the bowl classifier is returned to the concentrator by a 900-foot wooden launder, thence it is pumped by two 4-inch Wilfley centrifugal pumps to the tailings dam. The sands, averaging from 400 to 500 tons per day are fed by two 3-inch, gravity, rubber-lined pipe lines to the mine stopes to be used as fill. When the sands are not needed in the mine the mill tailings from the mill thickener are pumped directly to the tailings dam

SAMPLING

Three Geary-Jennings automatic samplers are located as follows:

No. 1.- Heads sampler; placed at the trommel undersize discharge; 15-minute cut interval.

No. 2.- Concentrates sampler; placed at discharge of No. 2 cleaner flotation cell; 15-minute cut interval.

No. 3.- Trial sampler; placed at discharge of No. 2 scavenger flotation cell; 30-minute cut interval.

All minor sampling is accomplished by the use of hand cutters.

WATER SUPPLY

Figure 8 is a flow sheet of the water supply.

When milling a normal tonnage of 1,300 tons per day, the mill consumes about 600 gallons of water per minute. The ratio of water to solids in flotation feed is roughly 2 to 1, or approximately 35 per cent solids. Due to dilution from various launder sprays and wash water also to the large tonnage of concentrates produced, the ratio of water to solids in the tails is 3 to 1, or 25 per cent solids.

Of the 600 gallons of water consumed per minute, 300 gallons are recovered from the tailings dam, 150 gallons from concentrates and tailings thickener overflows, and the remaining 150 gallons are made up of new water from the stream at the foot of the mill.

The water from the tailings dam is recovered by a single stage, double suction, centrifugal pump. It is collected in a reserve tank with the thickener overflow and fed to four 6 by 8 inch vertical triplex plunger pumps which send this water to two mill supply tanks above the concentrator. The new water is pumped direct to the mill supply tanks.

There is no treatment of the recovered tailings water for purification as it is allowed to settle and is exposed to the sun and air for some time before being used again. The water-supply cost is 1.4 cents per ton.

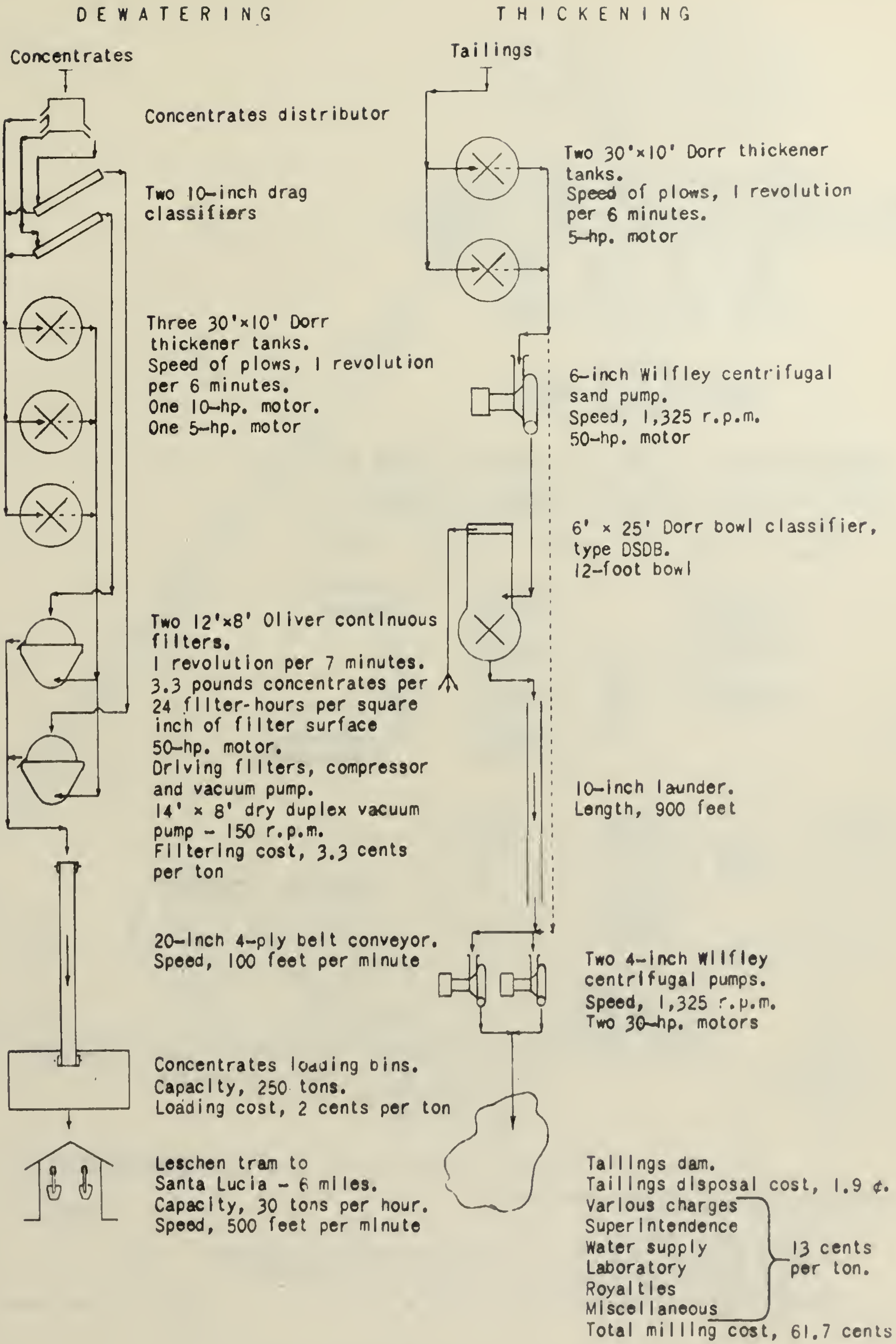

Figure 7.- Flow sheet of dewatering and of thickening

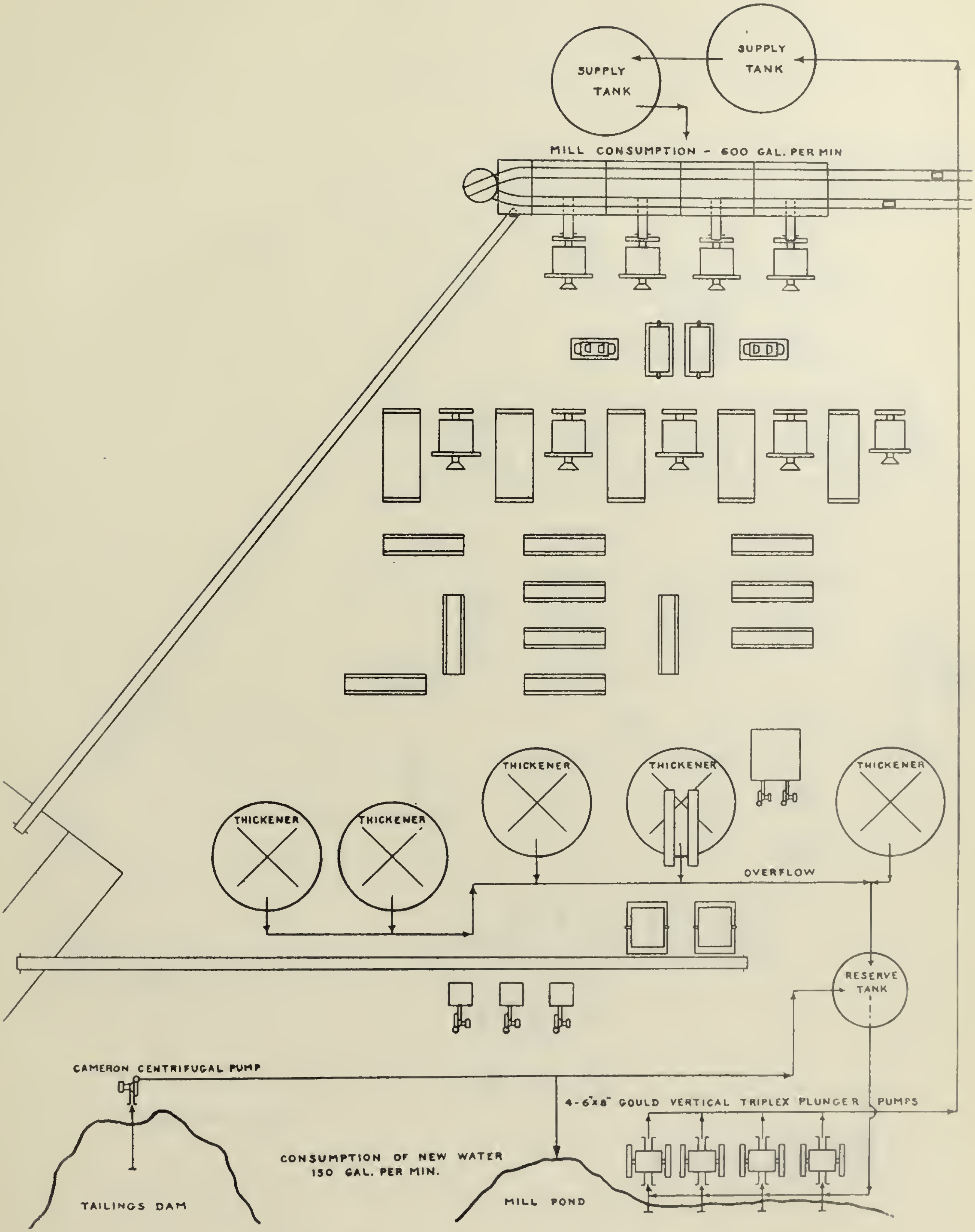

Figure 8.—Flow sheet of water supply

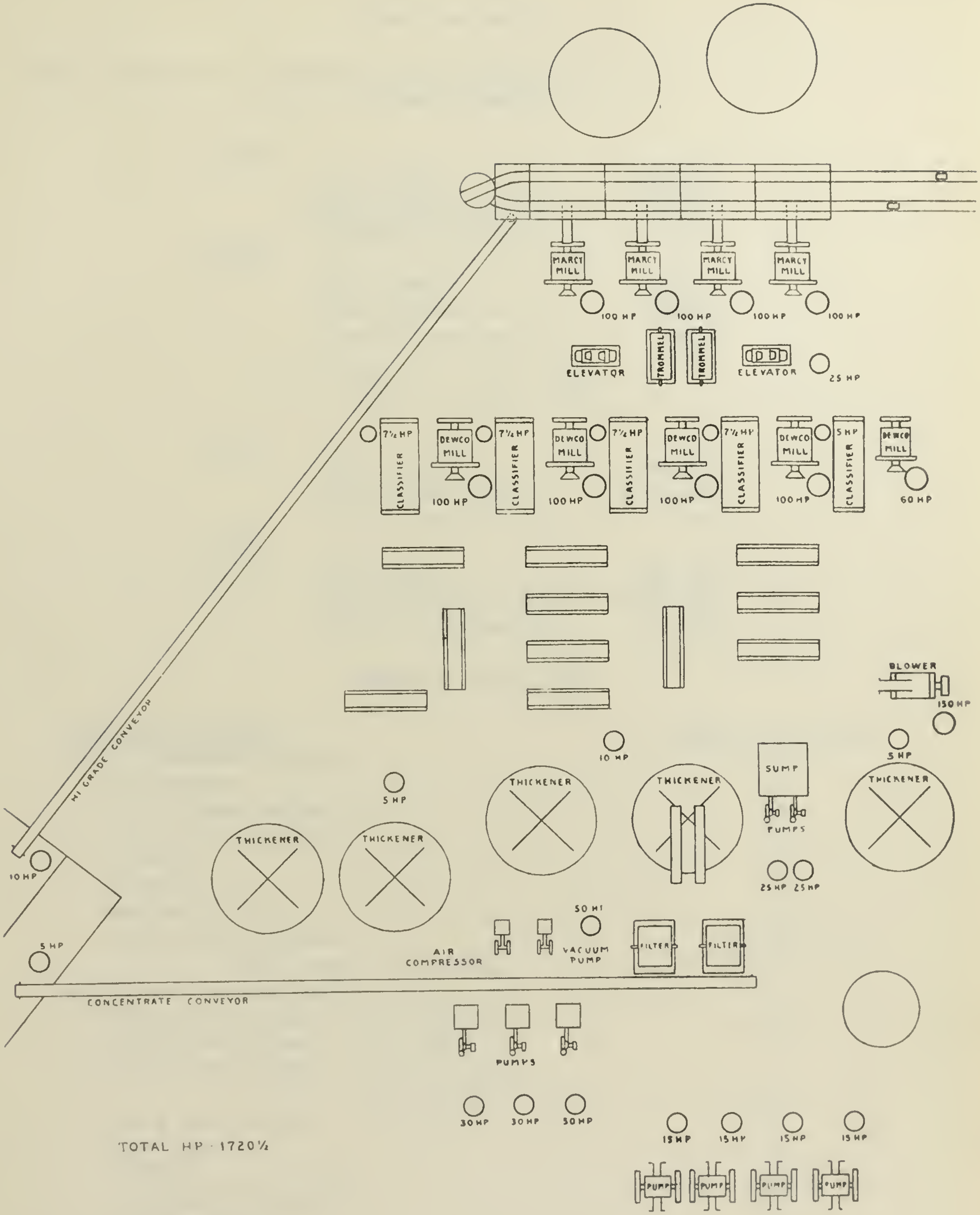

Figure 9.– Flow sheet of power distribution

POWER CONSUMPTION

Power distribution is indicated in the flow sheet Figure 9.

PRIMARY CRUSHING

1929:		
	Tons crushed	362,688
	Total kilowatt hours	929,578
	Kilowatt hours per ton	2.56
1930:		
	Tons crushed	365,374
	Total kilowatt hours	914,088
	Kilowatt hours per ton	2.50

MILLING AND CONCENTRATION

1929:		
	Tons milled	356,212
	Total kilowatt hours	5,577,467
	Kilowatt hours per ton	15.66
1930:		
	Tons milled	359,468
	Total kilowatt hours	5,653,392
	Kilowatt hours per ton	15.73

SUMMARY OF COSTS, 1930

Crushing and sorting	Cost per ton, cents
2 - No. 6 Kennedy crushers	2.7
Sorting belt	1.0
1 - 4-foot Symons crusher	2.6
1 - 54 by 20 Traylor rolls	3.8
Screening	.5
Conveying	1.0
Receiving bins	.4
Miscellaneous charges	1.0
Total crushing and sorting	13.0
Milling and Concentration	
4 - 64-1/2 Marcy mills	12.7
Elevator and trommels	1.1
4 - 6 by 6 Dewco mills) 1 - 5 by 5 Dewco mills)	13.6
Classifiers	.5
Flotation	15.4
Filtering and dewatering	3.3
Receiving bins	.2
Tailings disposal	1.9
Various charges	13.0
	61.7
Grand total - crushing, milling, and concentration -	74.7

I. C. 6545

November, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

THE BUREAU OF MINES COAL-SAMPLING TRUCK

BY

R. H. Kudlich

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

THE BUREAU OF MINES COAL-SAMPLING TRUCK[1]

By R. H. Kudlich[2]

In order to assist the various Federal and State agencies using large quantities of coal to select coal best suited technically and economically to their use, the United States Bureau of Mines has placed at their disposal a fuel inspection service. As part of this service, inspections are made at the mines which propose to furnish coal to these agencies on contracts. Such inspections cover both the mine itself to ascertain its capacity to furnish the desired tonnage and quality of the coal, the preparation of the coal in the tipple, and the obtaining of samples for analysis to determine the quality of coal as shipped.

To insure that the sample sent to the laboratory for analysis is representative of the average quality of the coal, increments are taken at intervals until a gross sample of about 1,000 pounds of coal has been collected from each of the sizes of coal shipped by the mine, while coal has been coming from all parts of the mine. In reducing this gross sample to the small sample which is sent to the laboratory for analysis it is necessary to crush and quarter the coal by a certain standardized method until only 5 or 6 pounds of coal small enough to pass through a 3/16-inch square-mesh screen remains. The ordinary method which the bureau has followed was to crush the coal by hand with a tamper, such as is used for tamping concrete, on some hare smooth surface and to reduce the volume by quartering and riffling.

During the past few years the demands from various Federal and State agencies for this service have been increasing rapidly in respect to bituminous coals. In addition to this, which might be called the normal increase, numerous requests have been received for similar service in respect to anthracite. It was obvious that if this increased demand was to be met it would be necessary either to increase the force allotted to this work or to provide the present force with some means by which their work could be made more effective. A careful survey of the situation showed that the greatest amount of time was consumed in transporting the men and their equipment from mine to mine in the same district, and in reducing the gross sample to the small finely crushed sample which is to be sent to the laboratory. Both of these phases of the work could be greatly accelerated by providing a motor truck of suitable speed and capacity, carrying a power-operated crusher, and the proper auxiliary equipment to store the gross sample as collected and to reduce it in volume efficiently after it has been crushed. Such a truck has been provided and placed in service. In the eight months it has been in operation the desired results have been fully attained. About twice as many samples as usual have been taken during that time by the same 2-man crew assigned to this work.

Many of the mines in the eastern part of the United States are in hilly or mountainous regions and can be reached only over rough, practically unimproved roads. Hence the specifications for the truck were drawn to cover high-quality, long-life trucks, in order that

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is usee: "Reprinted from U. S. Bureau of Mines Information Circular 6545."

2 Assistant to chief engineer, mechanical division, U. S. Bureau of Mines.

time in the field would not be reduced by frequent trips to the garage for repair and overhauling. The estimated total weight of the truck body and fixed equipment was 4,800 pounds and the accessory equipment 550 pounds. However, a "body and pay load" capacity of 7,000 pounds was specified, since when in service one or two gross samples of about 1,000 pounds each are loaded on the truck platform, and also when the crusher is in operation it may set up violent vibrations if a heavy piece of slate or rock should become caught between hammers. When fully equipped the truck with its 2-man crew and their personal baggage weighs somewhat over 10,000 pounds.

Figure 1 indicates the general arrangement of the truck and its equipment.

The truck body consisted of an underframe which serves as a bed plate for the crusher and engine and as a "backbone" for the truck body proper. The underframe is made of 6-inch steel channels, I beams and flats riveted and welded together to obtain maximum stiffness, the longitudinal members spaced to correspond with the longitudinal frame members of the chassis and the cross members to accommodate the crusher and engine. The body proper consists of a floor 6 feet 6 inches wide by 10 feet long with sides 20 inches high. The sides and tail gate are hinged, so that they may be lowered and supported in a horizontal position so as to form a platform approximately 10 feet wide and 11 feet 9 inches long. This gives ample working room and space in which to assemble two gross samples in the canvas bags in which they are collected. With the sides up, there is approximately 36 cubic feet of space available for carrying the auxiliary equipment, supplies, and personal baggage of the crew. While on the road, the entire body and equipment are covered with a fitted tent-shaped tarpaulin stretched over a ridge pole carried on two removable pipe stanchions.

A swing-hammer crusher fitted with 3/16-inch gratings was selected as having the proper capacity and crushing the coal to the fineness called for by the standard method for coal sampling. A welded sheet-steel hopper of proper size to receive the coal poured from the canvas bags has been added to the top of the crusher and a discharge hopper which fits into the riffle to the bottom. In order to conserve "head room" below the riffle, the discharge hopper was fitted up into the housing of the crusher as show in Figure 2.

The crusher has a rated capacity of 1,000 pounds per hour at a speed of 1100 r.p.m. It actually crushes a 1,000-pound sample of moderately hard bituminous run-of-mine coal in from 7 to 15 minutes and a 1,000-pound sample of $\frac{3}{4}$-inch slack in from 5 to 9 minutes.

The crusher is driven by a 20 hp. gasoline-engine power unit to which it is connected through a flexible coupling. The power unit is self-contained, consisting of a 4-cylinder 4-cycle gasoline engine with a forced-circulation water-cooling system, magneto ignition, self-starter, and with a clutch on the power take-off shaft. A mechanical governor holds the speed at about 1,200 r.p.m. Since a lot of dust is present in the air where the engine is to operate, the air inlet to the carburetor has been fitted with an oil-type air filter. The entire unit is enclosed in a weatherproof sheet-metal housing. The exhaust outlet in the top of the housing has been provided with an exhaust pipe of thin-walled steel tubing sufficiently long to discharge the products of combustion considerably above the breathing level of the man feeding the crusher. This pipe is made easily detachable to allow demounting when the truck is in transit.

A single-cylinder 2-inch diameter by $1\frac{1}{2}$-inch stroke air compressor, belt driven through tight and loose pulleys from the power unit, supplies compressed air at a 100-pound pressure for cleaning out the crusher after a sample has been ground in order to prevent contamination of the next sample. Air is stored in a steel tank 18 inches in diameter and 22 inches long to clean the inside of the crusher and to dust off the outside of the crusher and power unit when the outfit is being cleaned ready to pack up.[3] An unloading device prevents the building up of excessive pressure.

3 Several months' experience has shown that this tank should have been somewhat larger.

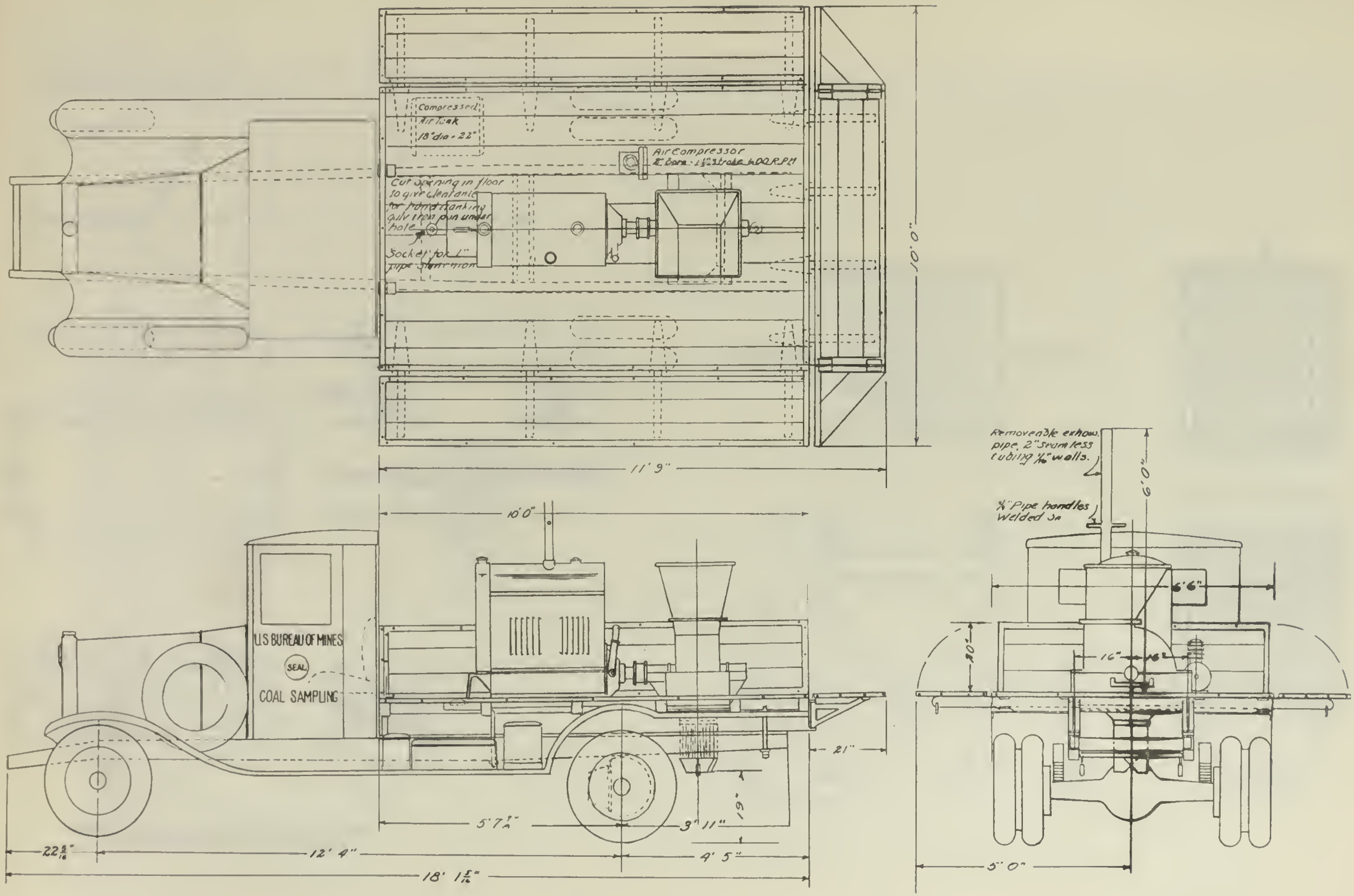
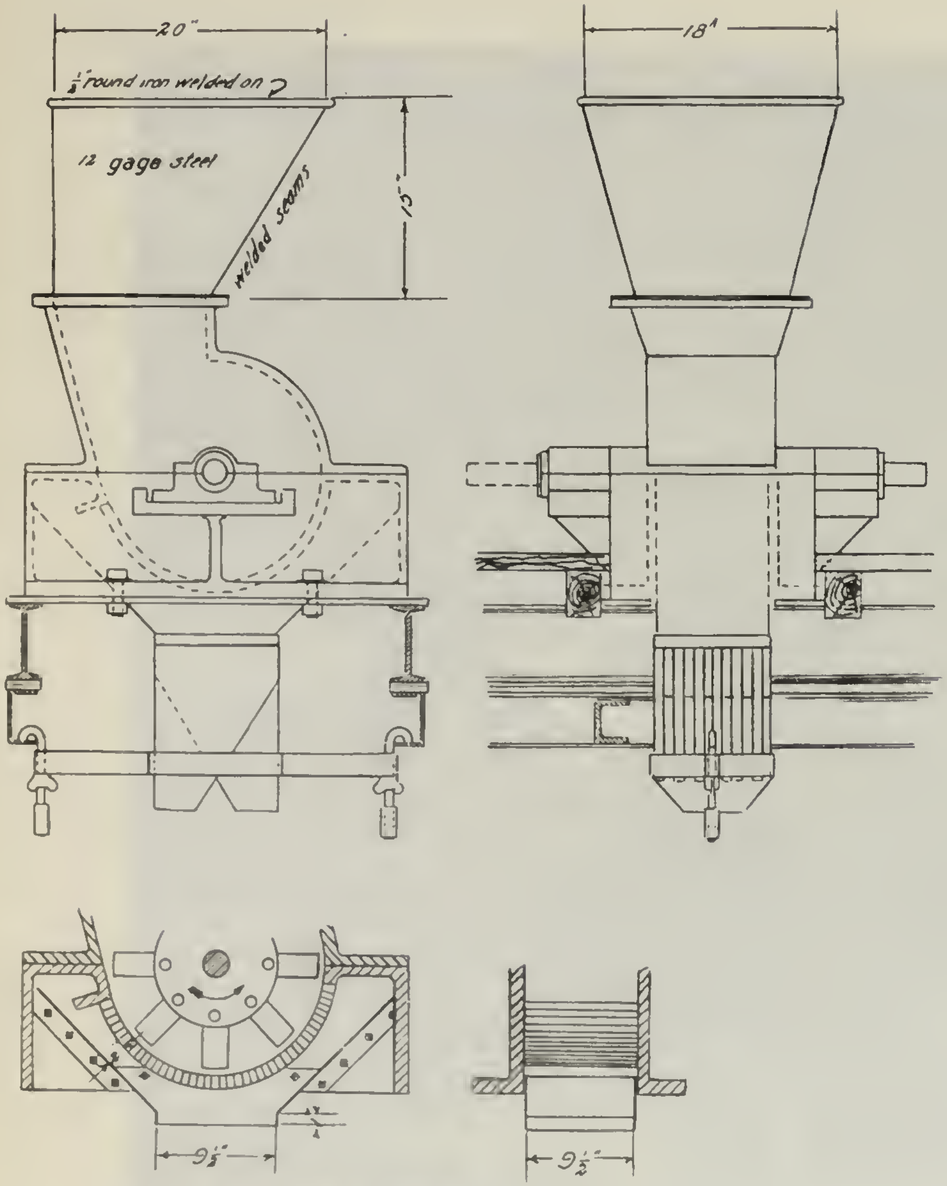


Figure 1.—General arrangement of truck



NOTE: Discharge hopper 16 gage galvanized iron to fit tightly inside of crusher housing

Figure 2.—Assembly of crusher, hoppers, and riffle

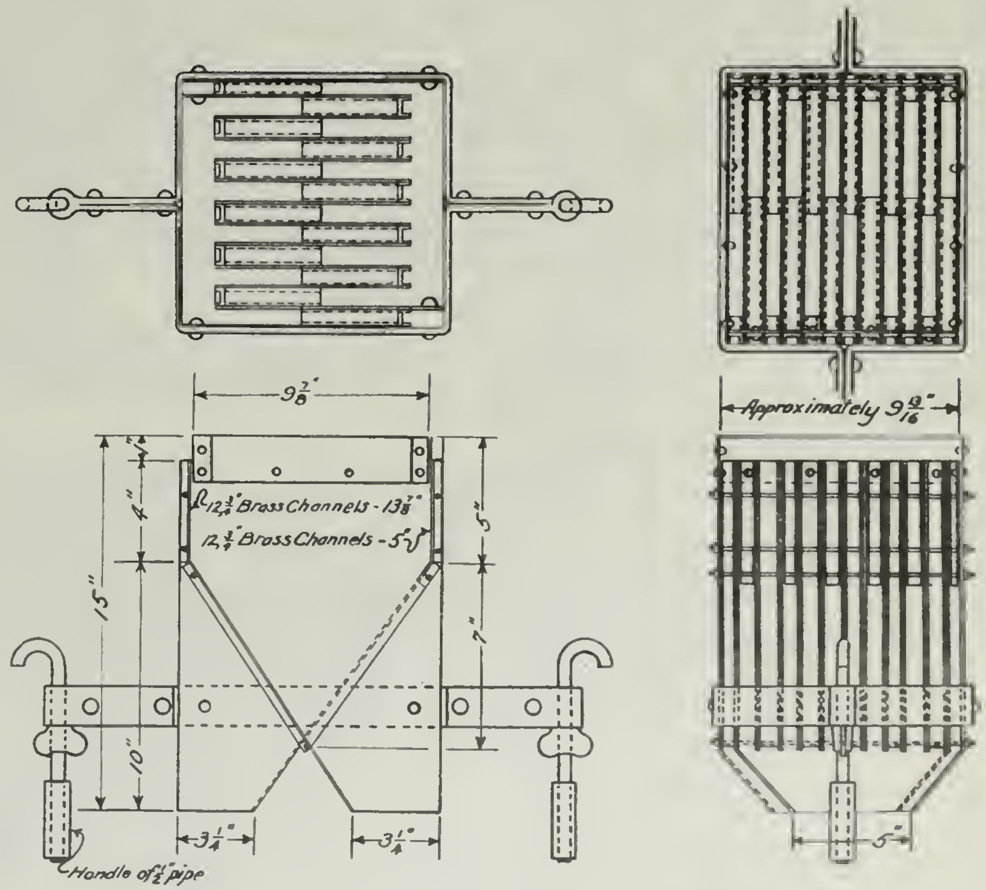
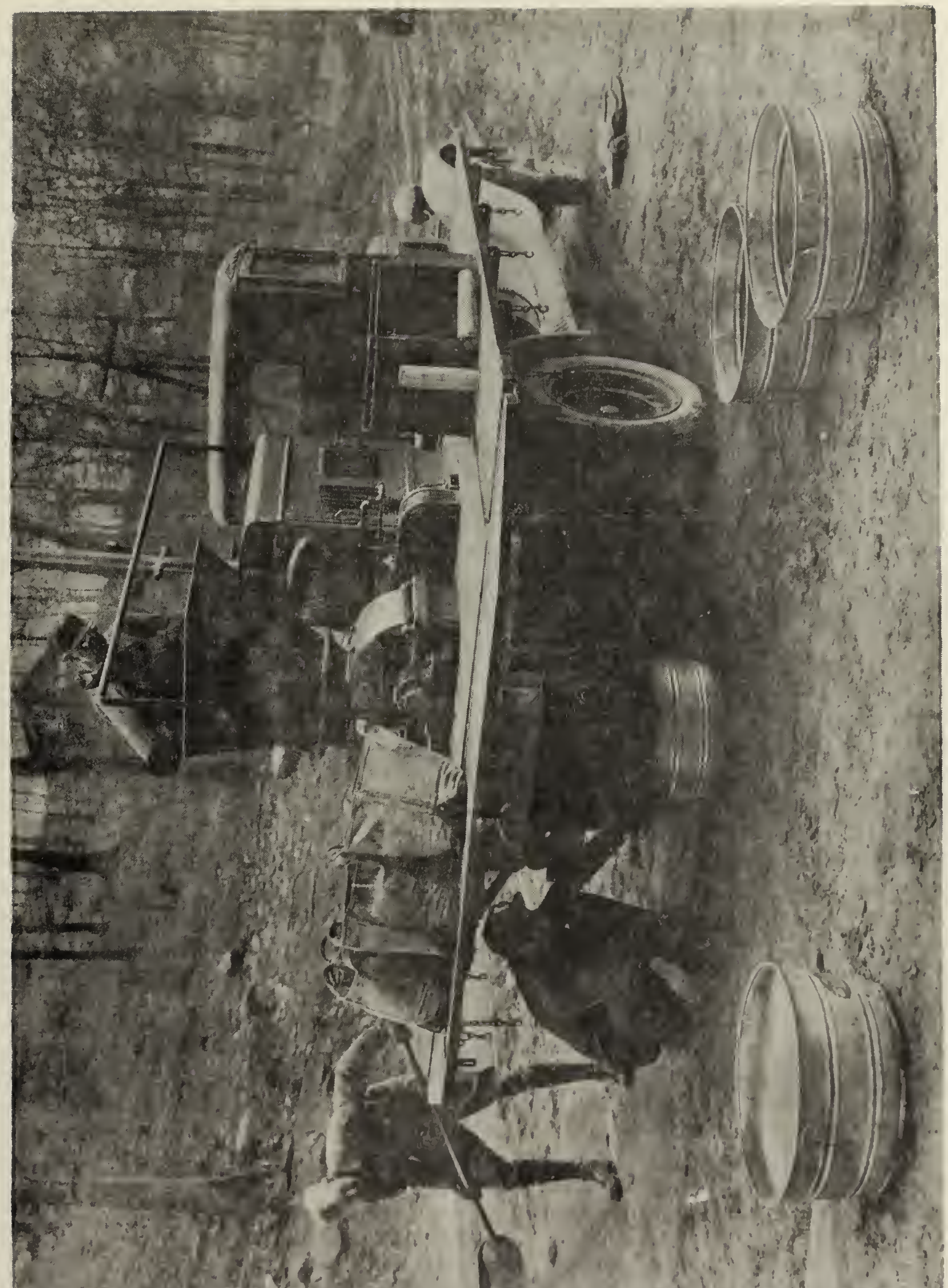


Figure 3.—Construction details of riffle



In order to reduce the manual labor required to reduce the crushed 1,000-pound gross sample to the 5-pound laboratory sample, the coal discharging from the crusher passes through a riffle attached to the discharge hopper of the crusher. This riffle constructed of brass plates and channels as shown in Figure 3 is designed to split the stream of coal into 12 equal streams; alternate streams pass to the right and left side of the riffle, the material which passes to the right being retained for further reduction and that to the left being discharged. The riffle when in service is supported on arms which hook on to the lower flange of the chassis frame. Since the accuracy of the splitting depends on having the riffle clean and the individual plates evenly spaced, the riffle is detached and carried in a suitable box when in transit to prevent its being mutilated or getting the slots plugged with road dirt. When the riffle is removed, the bottom opening of the crusher is protected against road dirt and moisture by a plate pivoted on the cross member just in front of the crusher and clamped to the one behind it by a thumb nut.

As auxiliary equipment the truck carries a pair of United States Bureau of Mines type riffle buckets for further reducing the retained half of the crushed sample; 55 canvas bags like those commonly used by coal dealers in delivering coal, into which the gross samples are collected; four heavy galvanized-iron wash tubs which are used in collecting, mixing and further reducing the retained half of the crushed sample; canvas covers for these tubs; and tarpaulins for covering the gross sample to prevent changes in moisture content and contamination of the sample; long-handled, square-point shovel, broom, hand hammer for breaking large lumps in the gross sample: a supply of the screw-top galvanized-iron cans in which the sample is sent to the laboratory; and a hand line for lowering the bags of coal from the tops of the railroad cars or loader's platform of the tipple.

The methods used in collecting the gross "hand sample" are used in sampling with the truck. A gross sample of about 1,000 pounds of each size of coal shipped is collected during the time it takes to "cover the mine" - that is, for coal to have been received at the tipple from each part of the mine - to insure that the sample be representative of all the coal that may be produced from the mine. It has been found that 1,000 pounds of bituminous coal fills about 12 of the canvas bags, and this number is apportioned to the number of railroad cars of each size loaded while the mine is "covered." As the samples are being collected the bags are grouped according to the size, and each group is kept covered with a tarpaulin to prevent contamination and change in moisture content.

When the gross sample or samples have been completed, each sample is passed through the crusher; one half of the sample is rejected and falls to the ground; the other half is collected in three tubs which are slid under the riffle spout in turn. Figure 4 illustrates the operations. The crushed coal in each tub is then thoroughly stirred and shovelfulls taken from each tub in turn to fill one riffle bucket, which is then dumped, one half of the contents being retained in the second riffle bucket, the other half being discarded. When the second riffle bucket is filled it is dumped back into the first, making the third "split" from the original sample. The retained portion is then transferred to the fourth tub, and this procedure repeated until the retained half of the original sample is disposed of. The portion retained in the fourth tub is then treated in the same way, splitting it three times until the retained portion has been reduced to about 15 pounds. From this the final samples which are to be sent to the laboratory are taken.

Though the procedure in reducing the gross sample does not follow literally the American Standard hand crushing and quartering method, it is quite analagous.[4] In the former method, after crushing, the sample is repeatedly mixed by piling in conical piles which are then

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4 See Pope, G. S., Directions for Sampling Coal for Shipment or Delivery, Tech. Paper 133, Bureau of Mines, 1917, 15 pp.

flattened out and divided into quarters, alternate quarters being retained. In the latter, the coal is thoroughly mixed in the tubs, shovelfulls are taken from different tubs in rotation, and alternate twelfths are retained. This method may be considered even preferable to the hand crushing method; less opportunity is offered for tampering with the sample, it is not exposed to the elements to so great an extent, and there will be less chance of gaining or losing moisture or being contaminated during the process, as is possible when the sample is spread in a thin layer over a large surface which may not be perfectly clean and may be exposed to wind-blown or falling impurities.

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December, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

SAFETY AT THE OLD DOMINION COPPER MINE, GLOBE, ARIZ.

BY

R. I. C. Manning and Albert Tallon

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFETY AT THE OLD DOMINION COPPER MINE, GLOBE, ARIZ.[1]

By R. I. C. Manning[2] and Albert Tallon[3]

The Old Dominion Co. at Globe, Ariz., like the Phelps Dodge Corporation with which it is closely allied, considers safety a major operating problem. Safety is fostered by this company in consideration of both the humanitarian and economic aspects; the officials take pride in promoting the welfare of employees, and the management has found that the safest way is the most economical and that unless an organization is operated economically it can not continue to exist successfully.

In planning and operating any safety organization, many things must be taken into consideration because an enterprise can not be operated solely for safety any more than it can for any other single part of the operations. All work must be planned and executed for the common interest.

In studying a safety organization and the results obtained by it, some knowledge of the conditions prevailing prior to its existence is necessary as well as a complete understanding of the present conditions under which it has to operate. Therefore, in perusing this report, it will be well for the reader to keep in mind certain handicaps existing at the Old Dominion mine, which are:

High rock temperatures.
High humidity.
High air temperatures.
Wide variance in stoping methods, brought about by various grades and conditions of ore deposits, making standardization difficult.

The following paragraphs regarding temperatures and humidity are taken from "Ventilation of the Large Copper Mines of Arizona," United States Bureau of Mines Bulletin 330, by G. E. McElroy, 1931.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6546."

2 Associate mining engineer, U. S. Bureau of Mines Safety Station, Salt Lake City, Utah.

3 Safety engineer, Old Dominion Co., Globe, Ariz.

United States Weather Bureau records for Globe, covering 26 years, show an average yearly mean dry-bulb temperature of 63.5°. No relative humidity or wet-bulb temperature data are available, but low humidities prevail and wet-bulb temperatures average about 15° lower than dry-bulb temperatures. These high average temperatures and high rock temperatures make the ventilation of the mine a difficult problem.

The range of temperatures at working places is wide and depends primarily upon the proximity of the working places to downcast shafts. Stope temperatures range from 70 to 90° dry bulb, with an average of about 80° accompanied by an average wet bulb depression of about 5°. Development faces are, as a rule, somewhat warmer, 80 to 95°, with only 2 to 3° wet bulb depression, but they have greater air motion than stopes and are comparable in comfort. Practically all places off the intake air courses are uncomfortably warm, the maximum condition of discomfort noted being on the 2600 level development drift off the A shaft where the air returning to the shaft at a velocity of 250 feet per minute was saturated at 102° F. Rock temperatures at the face of this drift were 108.5°.

PROPERTY

The Old Dominion property is at Globe, Gila County, Ariz., and consists of a mine, concentrator, and the necessary shops pertaining thereto. This copper mine is relatively deep, and operates on a series of steeply dipping parallel fissure veins which cut sedimentary limestones and quartzites intruded by diabase and covered with dacite. The mine is divided into the east and west sides. On the east, fairly regular bodies of sulphide ore predominate, whereas on the west the ore occurs as irregular bodies of oxidized and secondary ores cut by numerous faults. The average dip of the veins is 80° from the horizontal, and the strike east and west. The principal ores are chalcopyrite, chalcocite, and bornite with low-grade sulphides. Copper ore was first mined in this district in 1881.

For complete information on the geology and history of this mine see "Geology of the Globe Copper District, Arizona," United States Geological Survey Professional Paper 12, by F. L. Ransome, 1903. The ore production for December, 1930, was 40,000 tons averaging 2 per cent copper, and was mined by an underground force of 625 men.

The mine is opened by five vertical shafts -- namely, A, K, Grey, Pump, and C.

Operations are carried on through the A shaft, consisting of five compartments -- one for water pipe, two for handling men and supplies, and two for skips. A single-deck cage is permanently secured under each skip, and at the time the shift is changed, double-deck cages are fastened under these cages; thus, there are four triple-deck cages, with a capacity of nine men to a deck, available for handling the shift of men.

The other shafts are for ventilation and emergency exits. Ladders are installed vertically with platforms at every 20 feet.

Haulage levels are run at approximately 200-foot intervals, vertically. Intermediate levels for mining are run about midway between the haulage levels. There is no uniformity for spacing of raises, as requirements for different mining systems vary greatly.

Five methods of stoping are used at present: Rill, top slice, Morenci slice (modified), square-set, and shrinkage. The size of the orebody, the character of the hanging wall and footwall, the mineral content, and other factors determine the method of mining to be used. About 60 per cent of the extraction is by the modified Morenci slice-caving system.

The mine is ventilated on the pressure system by three fans each of 125,000 cubic feet per minute capacity, each delivering about 80,000 cubic feet of air per minute per fan, at present, at the top of the C, K, and Grey shafts. These fans are aided by large natural-draft pressures and booster fans underground. The principal return air courses are the A and Pump shafts and one compartment of the K shaft. The Pump shaft, which is used for ventilation and pipe lines, is vertical and is 100 feet west of A shaft.

The main fans are forward-curved, multiblade centrifugals. Those at the C and Grey shafts have housings for reversing the direction of air flow.

The air flow underground is approximately 760 cubic feet per minute per man.

For complete description of mining methods at this mine see "Mining Methods at the Old Dominion Mine, Globe, Arizona," United States Bureau of Mines Information Circular 6237, by A. H. Shoemaker, 1930.

Compressed air is blown for one hour in all headings after blasting, before anyone is allowed to enter.

HISTORY OF SAFETY DEPARTMENT PRIOR TO 1925

On January 1, 1914, the mine safety department was inaugurated by placing a safety inspector in the mine for the purpose of preventing accidents, his time being devoted to underground work only.

His duty was to make a regular inspection of and report on the condition of all travel ways, main exits, manways, shafts, and such other places as were used for emergency ways not frequently visited by others; to inspect all working places; to have the necessary precautions taken in all working places to avoid any accidents, and make a monthly report of the conditions found. In case of a serious or fatal accident, either day or night shift, he was notified and visited the place as soon as possible; made a thorough inspection of the conditions; collected all available information by getting the names of all the witnesses and a statement from each one regarding all conditions, so as to be able to make a full report of the accident.

At that time, as well as at all other times, it was necessary for the officials to take the lead in this movement if adequate results were to be secured. Warning and danger signs were posted throughout the mine; considerable work was done in guarding the tops of chutes and manways to prevent men from falling into them, heretofore the cause of many serious and fatal accidents. In most of the square-set manways staggered ladders were installed, and where the ladders were not staggered, trapdoors were placed. Trapdoors were also installed in all cribbed manways, the distance between them not exceeding 20 feet. All working raises, shafts, and winzes at the height or depth of 50 feet were supplied with blasting batteries, or switches connected to the electric-light wires, and delay-action fuses. Considerable improvements were also in safety conditions around the underground machinery.

To make this movement more successful, it was deemed best to interest the shift bosses by explaining to them that improved conditions meant better work and by making it clear to them that they could perform no better service than that of trying to show an indifferent companion his duty in performing his work in a systematic and intelligent manner, understanding that he had to use his brains as well as his muscles, and recognizing the responsibility placed upon him which he must assume.

To accomplish some of these objects, a system of bonuses for the shift bosses, based on their safety record, was inaugurated. Each shift boss working 500 shifts or more per month without any lost-time accidents was given a bonus of $10. If there was any lost time and it was less than 1 per cent of the total shifts worked, a bonus of $5 was given. This proved successful, as it encouraged them to supervise their men more closely. Every shift boss was entitled to this bonus each month, regardless of the number of accidents in the previous month. In this way they all started out with a clean sheet at the beginning of each month. There was also a form compiled each month with their names, number of accidents, lost time, and other items. This made them more careful because they did not care to see their names with a large number of accidents opposite them.

To get the men interested in the movement it was decided that three men should spend two days each in going through the mine with the inspector. After the first three men had gone through with the inspector for two days, they went back to their respective places, and three more were taken. They were shown the different workings of the mine and the conditions which should be maintained. Conditions were shown that would influence them to use their best judgment in keeping their own places clean and safe and to try to prevent their fellow workmen from being careless.

While going through the mine they were invited to offer suggestions, call attention to anything they deemed necessary to prevent an accident, and ask questions. They were also encouraged to offer suggestions to their bosses whenever they deemed it advisable and to report to them anything they considered dangerous. The bosses were also instructed to be courteous whenever any suggestions were being offered. This proved successful in educating both the bosses and men in safety. Some of the men upon going back would tell their bosses of the improved safety conditions in other runs, and as the bosses were keen to keep their runs at least as good as the others, they

became active in improving safety conditions. In most cases the men who had gone through were also careful to keep their own working places in good order so that the men following them could not find any fault with their work pertaining to safety. After several hundred had received this safety training the men thought it best to have all the runs represented at the same time in the form of a committee, and recommended that this be adopted: the officials adopted the recommendation, which was the beginning of the present committee system.

In the month of April, 1916, all of the surface plant was added to the safety department under the same inspector. This work began similarly to the underground work by talking safety to the bosses and men to win their confidence and get them interested in the movement. Many improvements were made by placing guards around machinery, removing exposed set screws and keys, and installing new runways for oiling purposes, putting handrails on old runways without them, and making other improvements. Later, the foremen of the various departments decided to select one of their own men each month to make an inspection of their department and report conditions, and to send a copy of the report to the inspector; this plan was continued until the present system of committees was organized.

The present system was organized and the first meetings were held in February, 1925. The organization was divided into two divisions, underground and surface, each division having a central committee and a workmen's committee. At a special meeting of all the superintendents and foremen the surface central committee of 9 members was chosen, and also the underground central committee, consisting of the superintendent and all the foremen. The surface workmen's committee is composed of a representative from each department, and the underground committee of a representative from each boss working 10 or more men. The workmen's committees were appointed by the safety inspector after a careful canvas of the employees to ascertain whom they would prefer to represent them, with the understanding that retiring committeemen, after serving a 6-month term, would elect their own successors.

Meetings are held each month and the minutes of the workmen's committee are read to the central committee at their regular meeting, when the recommendations are considered. The minutes of the central committee are also read at the workmen's meetings so that they will have an understanding of the actions taken on their recommendations.

Since this system was organized a few changes have been made to stimulate a greater interest in the work. It is the duty of the underground workmen's committee to make a weekly inspection trip through their respective runs, to notify their shift bosses of the conditions found, and to make a written report to the inspector. While making their inspection trips they found such a great difference between the men who had served on the committee and those who had not that they deemed it advisable to reduce the term to 3 instead of 6 months, to get the men through quicker so they will all have an understanding of what the safety department is doing for them. This change has been made, and it is now in effect.

To stimulate the interest among the foremen and bosses, the surface central committee has included all superintendents and foremen as members, with an invitation to all shift bosses to attend all meetings when convenient, and the underground central committee has included all shift bosses as members. The safety department is now represented by 78 members.

Accidents require special study. A repetition of similar accidents indicates that the underlying causes have not been studied thoroughly nor the correct action taken. It is the duty of the workmen's committee to investigate and report on all time-lost accidents. To assist in making these reports and to encourage a thorough inspection, the committeemen are supplied with a special accident report blank, which is filled out by them and delivered to the safety inspector.

At the first meeting of each new committee it is customary to have some of the officials present, who express the feeling of gratitude of the company for the support of the workmen toward this movement. This encourages the committeemen and makes them feel that they have the support of the management which seems to give additional zest to the work on which they are entering. A chairman and a vice-chairman is elected from the floor to serve throughout the term, with the inspector acting as secretary.

At first the recommendations started to come rapidly, and continued to do so for some time. Recently both the underground and surface committees were asked why there were so few recommendations coming in at present. Practically the same reply came from both parties that their boss wouldn't give them any chance. As soon as the boss is told about anything, the work is done and there is nothing to report. This was gratifying to the safety department, and it is hoped that it will continue. This attitude shows that the safety education is spreading and more interest is being taken in the movement. In addition to the list of recommendations recorded, many suggestions have been made by the workmen's committees whereby special investigating committees have been appointed and the work done before the next meeting; hence the matter never went before the central committees; also, the same results have taken place at the central committees' meetings.

Several of the recommendations presented by the workmen's committees were considered too great an undertaking for the central committees, and a special committee was appointed to take the matter up with the general manager, and the work then completed. Also, a number of recommendations have been mentioned to the bosses which they considered to involve too great an undertaking, and advised the committeemen to bring it up at the safety committee meetings. Some of the big jobs would probably not have been done until someone was injured, if it were not for the cooperation displayed at these meetings. The value of these recommendations is apparent when we consider that from February, 1917, to September, 1920, the workmen's committees presented 288 recommendations, of which 284 were adopted and the work completed. In addition to aiding safety, it has been proved to the central committees that many of the recommendations materially increased working efficiency. Except for a few changes, this system was in force until March, 1925.

PRESENT SAFETY ORGANIZATION

On March 14, 1925, the safety department was reorganized and a system installed similar to that of the Phelps Dodge Corporation.

Attitude Toward Safety

The idea that safety is an integral part of the operating activities of this property is impressed on every employee from the manager down. The safest way is considered the practical and most economical way of doing any job. This attitude is taken in regard to routine as well as to new projects.

The management has assured itself that every boss is an enthusiastic booster for safety first, for it believes that discipline and instruction starting at the top are paramount in initiating and maintaining a good safety record. Workmen can best be reached by their immediate superiors, and these men are the real backbone of safety.

That the management is in earnest when it states that accident prevention must come first has been impressed on all local operating officials; safety and efficiency go hand in hand and are inseparable.

Meetings, literature, safety bonuses (for bosses), competition, and instruction are used by the company as effective means of impressing the ideas of safety on the bosses and men. Special talks at underground meetings are given by bosses and men, and each employee is made to feel that he is an integral part of the meeting.

That everything must be done in a safe manner and that safety must not be sacrificed for speed is taught to all the men.

Work of the Safety Engineer

The safety engineer devotes his entire time to safety work and cooperates with all departments, both on the surface and underground. He usually spends the mornings underground and the afternoons on the surface making plant inspections or writing reports.

All safety work comes under his direct supervision, and he reports only to the manager. He meets with all safety committees and acts as secretary. The testing of safety devices also comes under his supervision and all men entering the employ of the company are trained and examined in first aid and various phases of safe work, regardless of their official capacity.

Committees

All safety work is brought up before a committee for discussion. Each branch or division has its own committee, in addition to which there are several committees whose work pertains to the company as a whole.

The Old Dominion Co. is a subsidiary of the Phelps Dodge Corporation, and the corporation committee of this corporation is at the head of all local

committees. This company also competes for the Phelps Dodge Safety Trophies which are awarded annually to the division having the best safety record in frequency and severity.

Corporation Committee.

Chairman, P. G. Beckett, vice-president and general manager in charge of operations.

Members: Branch managers.

This committee meets quarterly at the various branches in order. (Bisbee, Globe, and Morenci, Ariz.; Nacozari, Mexico; Dawson, N. Mex.)

Branch Committee or General Safety Committee:

Chairman: Manager.

Secretary: Safety engineer.

Members: Heads of departments.

This committee meets once a month in the manager's office to discuss the general safety work and means of obtaining the fullest cooperation of the various departments to prevent accidents.

Bosses Committee:

Chairman: Elected annually.

Secretary: Safety engineer.

Members: All bosses in both surface and underground departments.

Meetings are held monthly to discuss various phases of safety throughout the plant.

Department Committees (two underground, three on surface):

Mine Committee - East side.

Chairman: General mine foreman.

Secretary: Safety engineer.

Members: Division foreman and three workmen. The three workmen serve three months, after which new members are chosen.

Mine Committee - West side.

Same as east side.

Mine Surface Committee.

Chairman: Mine superintendent.

Secretary: Safety engineer.

Members: Three workmen. Members serve three months.

Concentrator Committee:

Chairman: Concentrator superintendent.

Secretary: Safety engineer.

Members: Three workmen. Members serve three months.

Miscellaneous Surface Committee:

Chairman: Mechanical superintendent.

Secretary: Safety engineer.

Members: Three workmen. Members serve three months.

The secretary of all committees is the safety inspector, who serves permanently.

Workmen members of committees are appointed by the chairman in consultation with the safety inspector, and serve for a period of three months.

All committees meet once a month and make a monthly inspection of the part of the plant they cover.

The department committees have authority to pass on all safety questions in their respective departments, and proceed as soon as possible to carry out any safety provision. Delay is not only dangerous, but causes the men to lose interest. Workmen members are consulted freely and urged to give their ideas, and other workmen on the plant are asked to call to the attention of the committee any dangerous condition. Questions which involve a heavy expenditure or affect the general policy of the company are referred at once to the general committee for prompt action. When the representatives of the management on the committees and the workmen can not agree on any specific question,this is referred to the general committee for final decision.

Foremen and bosses are given to understand that they are not relieved of responsibility for the safety of their men. The committees are intended to coordinate safety efforts, promote safety interest, and assist the bosses and foremen to make their working conditions safe, but the ultimate responsibility for accidents rests with the boss. Bosses are drawn into discussion of proposed changes to be made in the interest of safety.

When a committee has decided on a safety measure, it must be put through as quickly as possible, and the order should not be changed or canceled without first consulting the committee. Neither manager, superintendents, nor anyone else may change a ruling of a departmental committee without the approval of the general safety committee or the departmental safety committee concerned.

Rules are enforced or repealed.

Committees see that after safety appliances are provided, the foreman insists on their use.

Each of the underground safety committees takes two days for the inspection. A meeting is held on the second afternoon after the inspection, when both committees' records of the previous month, the accidents for the current month, and the comparison of accidents for the year to date are all read and discussed, after which the meeting is thrown open for the discussion of any new safety suggestions or recommendations that may be brought before the meeting.

While making the inspection, each place is graded and marked in some conspicuous place with chalk as to the conditions found and decided upon by the committee. The following letters are used for grading the working places:

E for places that are in excellent safety condition, and no suggestions are made.

G for places considered good, but one or more minor safety suggestions are made.

M for medium.

P for poor.

This grading has created considerable rivalry among workmen. It is continued with a follow-up letter to each of the shift bosses, commending them for the excellent and good safety conditions found on their runs, and calling their attention to the unsatisfactory conditions. As a rule, the surface inspections are made and completed in half a day.

Following inspections by these committees, meetings are held in which various phases of the inspection are discussed, and letter reports are written by the secretary. Copies of this report are sent to the general manager, mine superintendent, general foreman, bosses, and safety engineer. The reports contain lists of gradings with discussions and reasons, and comparisons with similar gradings in other sections.

Workmen serving on committees wear a safety button during the three months, and a safety watch fob at the conclusion of their term. These buttons and fobs are presented by the company. Men take pride in serving on committees. This system serves as a valuable means of interesting the workmen, and creates large numbers of safety inspectors throughout the plant.

Each boss holds a meeting once a month underground with all of his men. The time and place is left up to the boss. The length of this meeting is usually about 20 minutes. A chairman and secretary are elected by the men. Discussions are held on various safe methods of mining, with particular emphasis on their own section. The safety engineer attends these meetings when possible, and usually gives a short talk.

In addition to the committee mentioned, there is a grievance committee, the membership of which consists of four men; one from mining department, one from mechanical department, one from concentrating department, and the company claim agent.

All of these men, with the exception of the claim agent, are workmen elected from their respective departments to serve one year. Bosses and salaried men are not eligible to vote or to serve on this committee. Meetings are held once a month and reports are sent to the general manager.

This committee also has charge of a relief fund for needy workmen and their families, into which fund workmen voluntarily pay 25 cents a month, and the company donates an amount equal to the entire workmen's contribution.

Hospital and Doctors

Single men pay $1.35 a month and married men $1.80 a month as hospital fees, for which they receive free medical and surgical treatment on prescription of one of the company's doctors, including drugs, dressings, and hospital for any disease or injury that can be directly attributed to their work. General practitioner services are also rendered free to the families of married men with certain exceptions, and in some cases services are rendered at cost.

Bonuses.

The mine supervisory force, including general mine foreman, division foremen, and shift bosses, receive bonuses for safety records as follows:

General foreman ... $55 for clear record each month.
Division foreman .. 50 for clear record each month.
Shift bosses ...... 25 on basis of accidents per 1,000 shifts. Bonus can go over or under this amount.

There is also a sliding scale of bonuses in effect, based on the percentage of time lost to total shifts worked:

Record	Foremen	Shift bosses
Clear	$50	$25
Nor more than 1/2 per cent	40	20
1/2 to 1 per cent	30	15
1 to 1-1/2 per cent	20	10
1-1/2 per cent or more	0	0

All records start anew each month.

Educational Work

Safety posters obtained from national organizations and also made locally are conspicuously posted on neat bulletin boards at strategic points on the surface.

A course in first aid comprising five lessons of two hours each is given to all new employees by the personnel of the Globe-Miami Mine-Rescue Association. Classes are held at times convenient for the workmen, regardless of the shift on which they might be working, and they are required to complete this course in six weeks or be discharged as a penalty for noncompletion.

At the employment office each man is given a lecture on the rules and policy of the company pertaining to safe work. He is also given a good for which he signs a receipt agreeing to read and abide by the rules contained therein. This book is of a size that is conveniently carried in the pocket and has a substantial binding.

The contents of the book are a forward by the manager, stating the general policy of the company, and a total of 23 rules under the following headings in order:

1. Safety buttons and certificates.
2. General hospital rules.
3. General rules.
4. Goggles.
5. First-aid boxes and tool kits.
6. Shafts and cages.
7. Timbering.
8. Explosives.
9. Electric equipment and haulage.
10. Materials and tools.
11. Holes and openings.
12. Loading from chutes and tramming.
13. Drilling and shoveling.
14. Lights and lamps.
15. General rules to be observed for mine fires.
16. General rules for surface departments.
17. Boiler and engine room.
18. Wood working and all shops.
19. Saws.
20. Jointer.
21. Boring machine.
22. Railroad.
23. Miscellaneous.

In addition to these rules there is a list of special rules for bosses which is included in the appendix.

Penalties. The penalty imposed for violation of any of the rules contained in the book is left entirely to the boss, and in some instances to the safety committee. There are no set penalties for the violation of any rule.

Safety Buttons and Certificates

1. When an employee of the Old Dominion Co. has completed six months service without a time-lost accident he will be entitled to a safety button. At the end of two years without a time-lost accident a different button will be awarded, and at the end of 5 years without a time-lost accident, a gold and enameled button will be awarded. With the button a certificate will be issued showing the length of time the employee has worked without a time-lost accident. Any man leaving the employ of the company will also be entitled to a certificate showing the length of time he has worked without a time-lost accident; to obtain this certificate he should make application for it to the employment foreman.

2. A time-lost accident is defined as one which causes loss of time after the day in which the accident occurs; 150 days worked will be considered as 6 months, 300 days as 1 year, 600 days as 2 years, and 1500 days as 5 years. Time off will not be counted.

3. An employee who causes an accident to a fellow employee which results in lost time beyond the day on which the accident occurred will be considered as having had a time-lost accident, even if he himself was not injured.

Physical and Mental Qualifications

Some types of work in the mine require certain physical and mental qualifications in order that safe and efficient performance of the same may be assured, such as grizzly work, blasting, etc. With this idea in view, new men are placed on types of work most suitable to their qualifications, as far as possible.

Equipment

The day before starting work, all new men must provide themselves with the following accessories, which are compulsory: hard hat, goggles, carbide lamp, carbide can, matches in moisture-proof container, and safety shoes.

The company charges for the lamp and the hard hat at cost. Credit for these may be obtained and is deducted from the first pay check. Goggles are furnished free.

The wearing of loose or unnecessary clothing is discouraged.

The cager has instructions to prevent anyone from entering the mine unless he is provided with the compulsory equipment; these instructions are carried out thoroughly.

Special equipment such as wrenches, tool sacks, powder sacks, fuse sacks, foot protectors, respirators, etc., is furnished if needed, before a man

goes to work. Every man is provided with a wrench before going to work, and he is charged for it. When he severs his connection with the company he returns this and receives his deposit.

All men performing work likely to produce flying particles are required to wear goggles.

1. Goggles must be used when chipping with either hand or pneumatic tool. This applies to cleaning as well as cutting metal.

2. Goggles must be used on all grinding with emery wheels, except in the case of grinding plungers and rods with the regular lathe equipment.

3. Goggles must be used when "flogging" tools are held in such a position that a chip or flake might fly upward.

4. Goggles must be used when handling or pouring hot babbitt or lead.

5. Goggles must be used on lathe work when the material being cut is hard or brittle and the cuttings have a tendency to fly.

6. Goggles must be used when there is danger that sawdust and other flying particles may get in the eyes.

7. Goggles must be worn when breaking rock or concrete, unloading lime or while doing any work which endangers the eyes.

8. Goggles must be worn while breaking bowlders, starting holes, picking or moiling hard ground, barring down ground, blowing out holes, barring down chutes from manways, and in all other work where there is danger from flying objects or dust.

Standardization of Work

Because of the several methods of stoping, standardization of all work has been difficult, but certain classes of work such as timbering in development, blasting, haulage, and development have been standardized throughout the mine so that if a man is transferred he will have identical working conditions as far as possible.

Following is an example of equipment furnished and instructions given to men working on grizzlies:

8-pound hammer with one end drawn to a pick point.
Safety belt with rope attached.
Pick and shovel.
Draw hook to turn rock over without using hands.
2 pointed bars, one 4 feet and one 6 feet long, with a loop handle to prevent smashing the hand.

New men are acquainted by the shift boss with the following instructions:

1. To use goggles when sledging.

2. Not to work one draw raise when the opposite one is hung up.

3. To use standard equipment properly.

4. Not to work without safety belt and rope attached.

5. How to make a primer.

6. How to prepare a "bomb" to place on a stick for blasting hung-up chutes.

7. How to operate blasting signals to notify workmen on haulage level below.

8. How to blast.

SAFETY DEVICES IN THE MINE

Drawings in connection with new work for the entire plant are examined and approved for safety and efficiency by the superintendent and head of the department.

Many safety devices have been standardized in the Old Dominion mine, most of which are foreign to other properties. The most outstanding of these are listed below:

Blasting. - Both the explosive and detonator magazines are of the dugout type, naturally barricaded, and at least 1,000 feet from any building. Electric lights are installed in gasproof containers, and the switches are so arranged that when the door is shut the power is off. An attendant spends his entire time at these two magazines.

One day's supply of explosives is transported underground daily at 9:00 a.m. in the original boxes and hand-trammed from the shaft to the underground explosive magazines, from which it is issued in canvas sacks to the miners.

Underground magazines do not contain more than one day's supply of explosive. They are electrically lighted with globes in gasproof containers. Detonator and explosive magazines are separated by at least 50 feet of solid ground.

About 90 per cent of the blasting is done electrically. Delay detonators ranging from 0 to 15 and of 8X strength are used. Power is 250 volts direct current and is taken from the trolley line. Fuse blasting is employed only in some of the square-set stopes.

Each working place has its individual blasting switch and line. Two wires are run to the working face. The permanent line is kept far enough from the scene of blasting to prevent damage to it. In the stopes all blasting switches are at the entrance. These switches are not locked, but

have a sign hung on the handle signifying their use. Gaps or interrupters are always in the blasting line between the switch and the face.

After the holes have been loaded, the detonator wires are connected in parallel, and to a temporary blasting line. This line is unrolled and supported, as the miners retreat from the face, and is attached to the permanent blasting line at a safe distance from the scene of blasting. The interrupters are then connected and blasting is done from the blasting switch.

The blasting switch is of the square, D type, encased in a steel waterproof box painted red, the lid of which has a baffle plate so arranged that the switch can not be closed unless the box is open. Coil springs attached to the switch keep it open unless it is held closed.

All men are instructed in the blasting procedure by the bosses.

Explosives have red wrappers and tamping has white wrappers.

Safety Belts.- Safety belts with ropes attached are used in all stopes, at grizzlies, and around shafts and pockets. The ropes are short enough to prevent a man from falling through the grizzly.

Grizzly Covering.- All grizzlies in the stopes are kept covered, when not in use, by two 2 by 12 inch by 6-foot sound planks laid side by side.

Drilling.- All drilling in the mine is done wet. Stoper handles have steel guards so attached that they will protect the hand from falling rocks.

Track Switches.- Track switches leading off the main-line haulage are equipped with air whistles and a red light so arranged that when the switch is thrown against the main line the whistle blows constantly and the red light remains on (see fig. 1).

Ladders.- Ladders are installed in raises, shafts, and other places, with lattice doors at every 20 feet. Lattice doors made from short lengths of ladder are used so that ventilation will be obstructed as little as possible.

A light, rigid steel extension ladder is used to repair chutes and raises. The extensions are $5\frac{1}{2}$ feet long. This ladder has many advantages over the chain or rope ladder--for instance, it has a better hand and foot hold (fig. 2).

When repairing chutes, a clevis is used for lowering or raising timbers. A hole is bored through the timber and the clevis attached by means of a bolt (fig. 3).

Trolley-Wire Guards.- Trolley wires are installed 7 feet above the rail, and box guards are used in front of chutes or where men work near them or pass under them constantly. This guard completely covers the wire. It has a tapered slot at each end. The trolley pole spreads this guard when passing, and the guard closes automatically by its own weight after the trolley has passed (fig. 4).

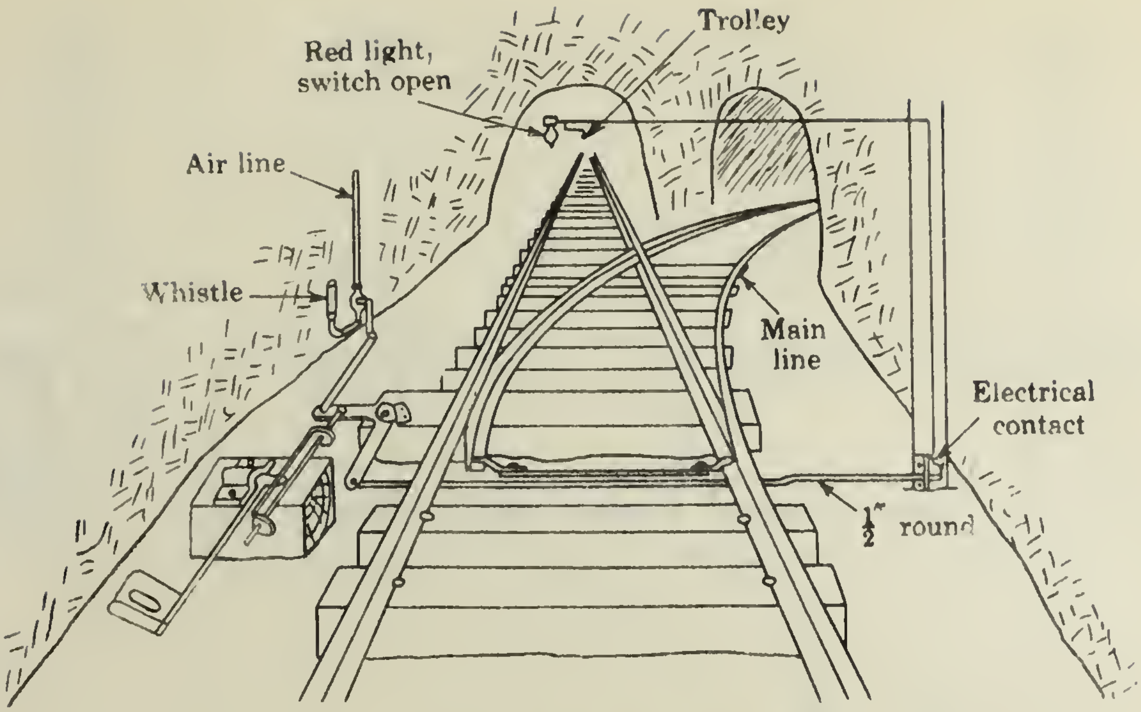

Figure 1.- Whistle and signal-light system

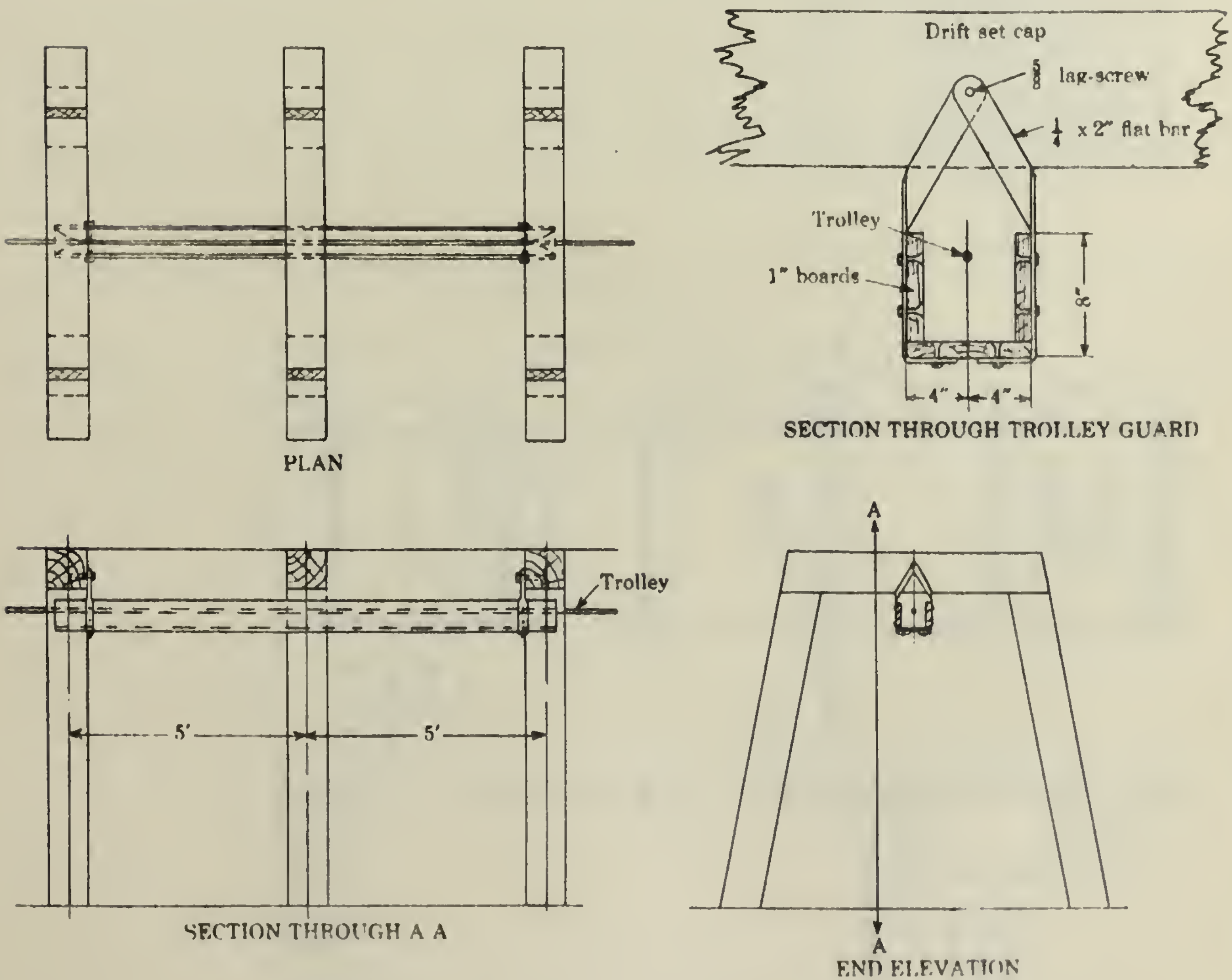

Figure 4.- Trolley guard

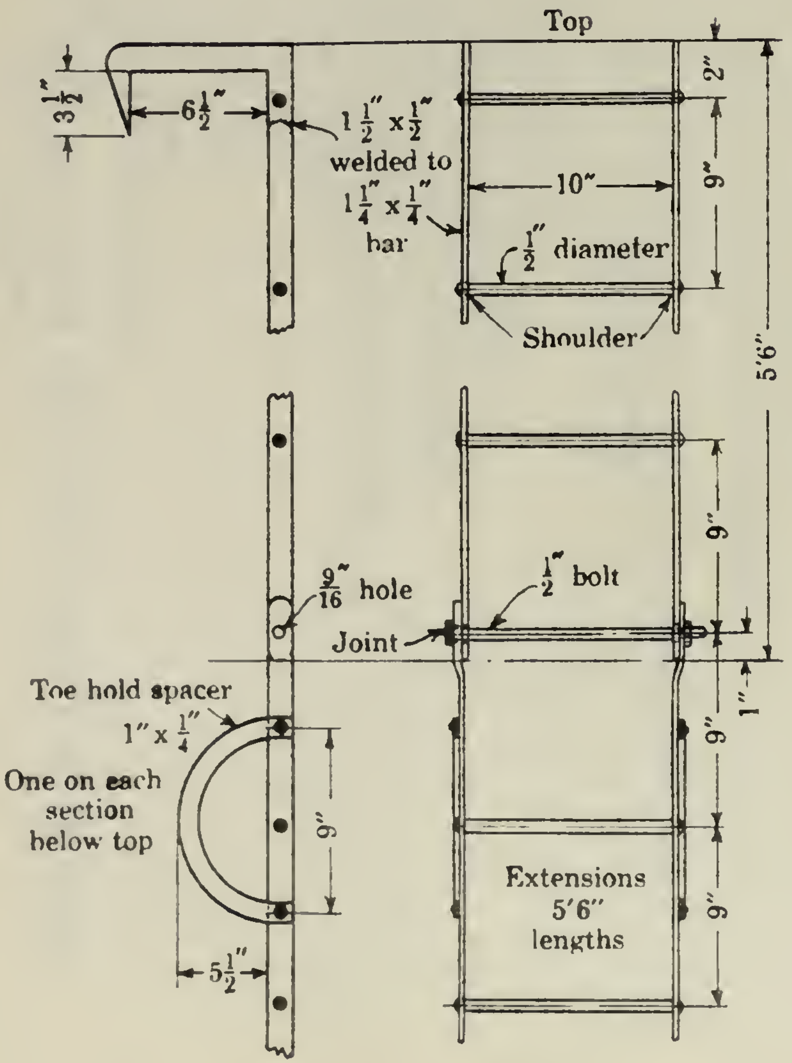

Figure 2.- Steel extension ladder for repairing chutes and raises

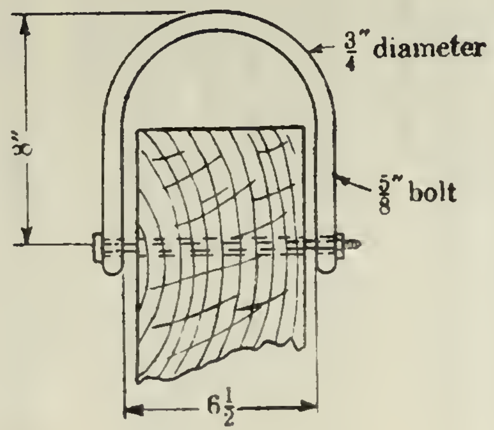

Figure 3.- Six-inch clevis used in repairing chutes

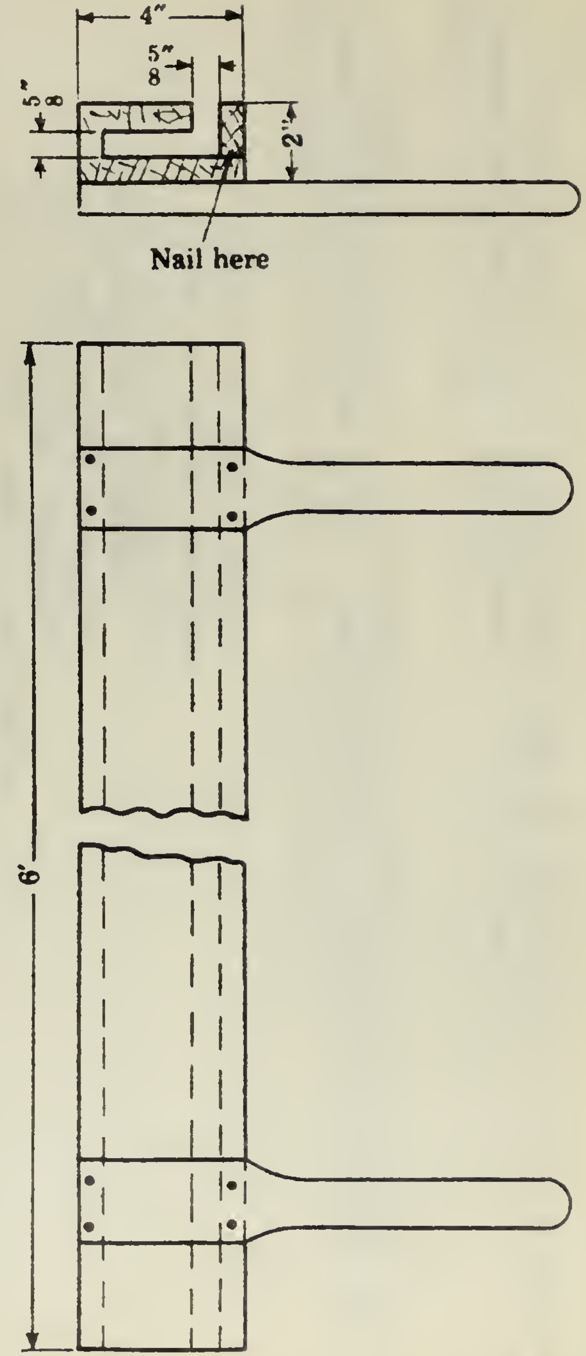

Figure 5.- Portable wooden trolley guard used by repairmen

Small portable wooden guards are hung on the trolley wire by men performing temporary work. These guards have to be removed before the trolley can pass over the wire (fig. 5).

First-Aid Material.- All employees carry adhesive mercurochrome compresses and ammonia inhalants. Large first-aid cabinets are kept at strategic points underground and contain the following materials:

1 bottle of picric acid.
Splints for leg and arm.
1 pound of cotton padding.
18 triangular bandages.
12 1-inch roller bandages.
12 2-inch roller bandages.
24 1 to 4-inch sterile compresses.

Long leg splints are used in place of the regular broken-back splint.

Trash Disposal.- One man spends his entire time cleaning up around the shaft stations and other places where trash accumulates.

Fire Prevention.- All workings are equipped with water lines for fire fighting, and enough hose connections are made for regular air hoses, since there is always a supply of these on hand, and miners are familiar with their use. The air line is also so arranged that water can be turned into it in the event of an emergency. In addition to this, fifty-fix $2\frac{1}{2}$-gallon capacity soda-acid fire extinguishers are stored at strategic places underground. Pyrene extinguishers are used on the surface.

Miners can be warned by means of a stench system attached to the air line and also by signals on the light line.

Regular fire drills are held on the surface.

Emergency boxes containing the following equipment are kept on all levels, several being on the larger levels:

1 4-pound hammer.
1 18-inch stilson wrench.
1 24-inch stilson wrench.
1 handsaw, 5 teeth to the inch.
1 claw bar.
6 double connections for air hose.
12 rubber gaskets.
1 wood chisel.
1 cold chisel.
6 ladder hooks.
24 pipe staples.
4 pounds assorted nails.
1 light axe.
Bell reducers $\frac{1}{2}$ to $\frac{3}{4}$ inch, $\frac{3}{4}$ to 1 inch, 1 to $1\frac{1}{2}$ inches $1\frac{1}{2}$ to 2 inches.

All fire equipment is regularly inspected.

REPORTS OF ACCIDENTS

All injuries, whether they are serious or not, are reported ~~by the bosses~~, on regular forms.

If the accident is serious enough to require medical attention, the boss issues a blue card to the injured workman which he presents at the hospital. This cardis returned to the general foreman by the doctor, and is later given to the shift boss when the man returns to work.

If the workman fails to report to his boss and goes to the hospital for treatment, a red card from the hospital is sent to his boss when he returns to work.

A man reporting to the hospital must be formally released before returning to work. Reproductions of the cardsfollow (fig. 6).

FIRST-AID AND MINE-RESCUE FACILITIES

The company is a member of the Globe-Miami Mine-Rescue and First-Aid Association which maintains a station 1 mile from the mine. At this station first-aid and mine-rescue apparatus is kept in readiness at all times for emergency use. Trucks are kept equipped with this apparatus so that there is a minimum of delay in getting to the scene of a disaster.

The personnel of the station consists of a director and three assistants, one of whom is on duty at all times. First-aid and mine-rescue training is given to employees of the Old Dominion Co. by the station personnel.

In 1930, all employees of this company were trained in first aid by a representative of the United States Bureau of Mines, and since then all new men have been trained in first aid by the personnel of the Globe-Miami Mine-Rescue and First-Aid Association.

A crew of 15 men has been trained in mine-rescue and recovery operations and receives additional training every month. There are also crews available on short notice from other mining companies in the district. These men receive a bonus of 30 cents a day.

A copy of the examination given these men is included in the appendix.

Tables 1, 2, and 3 give the accident statistics for the Old Dominion Co. and the Phelps Dodge Corporation, and Figure 7 shows the lost-time accidents over a 7-year period.

ACCIDENT ON DUTY

ACCIDENT CERTIFICATE FOR MEDICAL ATTENTION

OLD DOMINION COMPANY

Date............................ Hour..........

Old Dominion Dispensary (or Hospital).
Globe, Arizona.

The bearer,..No.........
is entitled to receive medical attention on account of an alleged injury received
While working for the Company on the........day of.............. 193...
at.............. P. M or (A. M.)

....
Time keeper, Foreman or Shift Boss.

This card must be presented at the Dispensary or Hospital before the expiration of 24 hours from the time of accident and returned to the General Office.

Received at O. D. Dispensary (or Hospital), Date.................. Hour.............

By...

Blue card issued by boss to injured workman

ACCIDENT ON DUTY

ACCIDENT CERTIFICATE OF MEDICAL ATTENTION

OLD DOMINION COMPANY

Date.................. Hour..........

Old Dominion Company
Globe Arizona

.... Shift Boss

The bearer...No..............
has received medical attention on account of an alleged injury while working
for the Company on the..........day of.................. 19.... at..........M
Probable loss of time..

By.. M. D.

Red card issued by hospital to injured workman

Figure 6.- Medical attention certificates

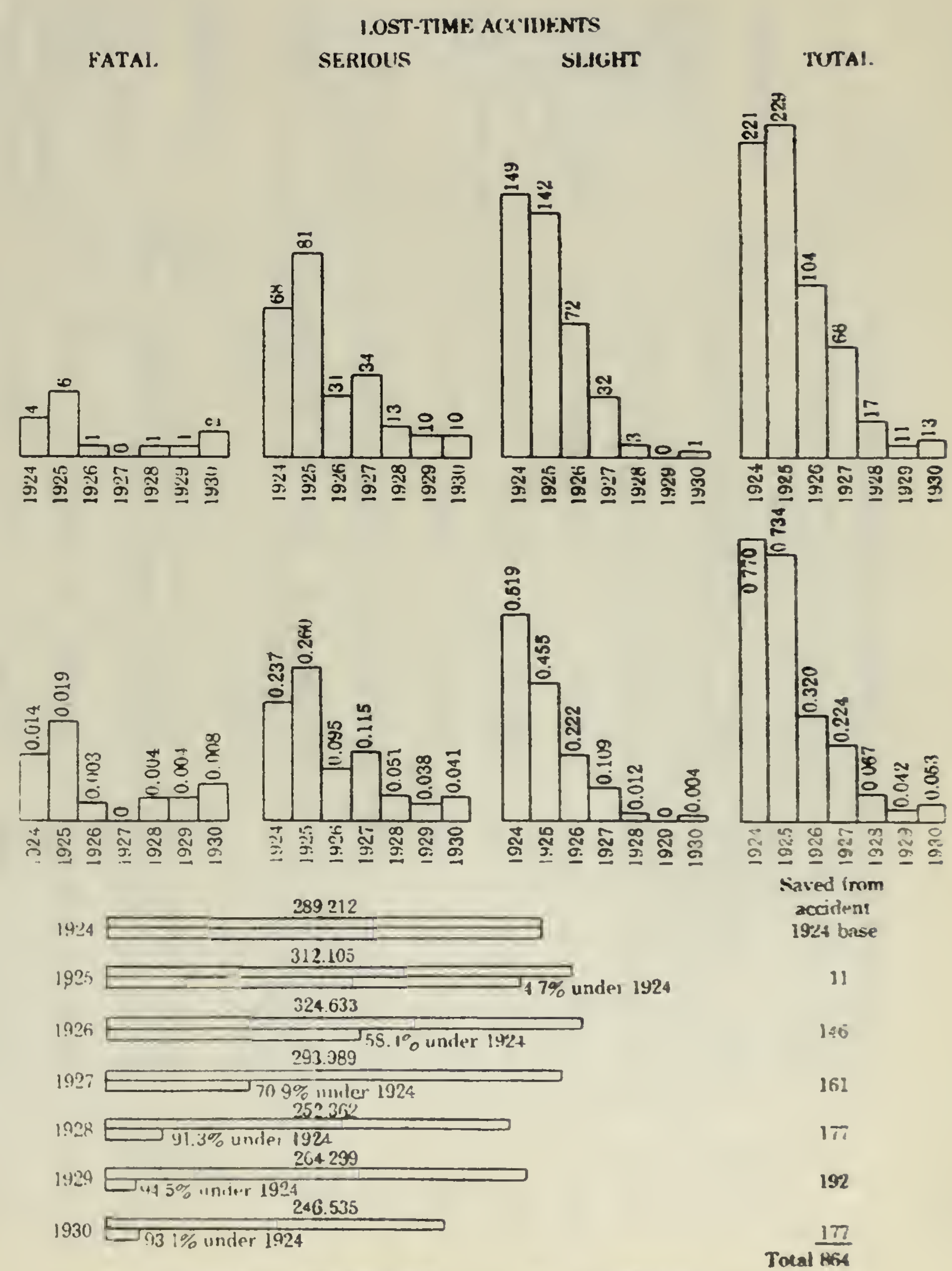

Figure 7.- Accident chart of the Old Dominion Co.

Table 1. - Lost-Time accident record of the Old Dominion Co.

Department	1930			1929			1928		
	Number	Total shifts	Per 1000 shifts	Number	Total shifts	Per 1000 shifts	Number	Total shifts	Per 1000 shifts
Underground mining	10	144,158	0.069	10	163,182	0.061	15	153,006	0.098
Crushing and concentrating	0	21,186	.000	1	20,500	.049	0	20,303	.000
Mechanical and shops	2	54,810	.036	0	52,813	.000	2	51,269	.039
Miscellaneous surface	1	13,538	.074	0	14,291	.000	0	13,179	.000
All other	0	12,843	.000	0	13,513	.000	0	14,605	.000
Total	13	246,535	0.053	11	264,299	0.042	17	252,362	0.067

Department	1927			1926			1925		
	Number	Total shifts	Per 1000 shifts	Number	Total shifts	Per 1000 shifts	Number	Total shifts	Per 1000 shifts
Underground mining	58	186,307	0.311	86	209,222	0.411	204	198,231	1.029
Crushing and concentrating	6	21,620	.277	2	22,273	.090	2	22,452	.089
Mechanical and shops	1	55,836	.018	14	58,502	.239	18	55,376	.325
Miscellaneous surface	1	13,819	.072	2	16,498	.122	5	18,146	.276
All other	0	16,407	.000	0	18,138	.000	0	19,900	.000
Total	66	293,989	0.224	104	324,633	0.320	229	312,105	0.734

Table 2. - Departmental Lost-time accident records of the Old Dominion Co.

Department	Present record			Best record		
	Date last accident	Days clear	Shifts worked	Successive days clear	Shifts worked	Dates
Underground mining	12/9/30	22	9,000	237	101,300	10/25/29 to 6/20/30
Crushing and concentrating	12/3/29	393	22,777	745	41,700	11/27/27 to 12/3/29
Mechanical and shops	8/15/30	138	18,574	713	103,300	1/21/28 to 1/5/30
Miscellaneous surface	9/17/30	105	3,159	1,229	50,500	5/5/27 to 9/17/30
All other departments	None since 1924	2190	103,065	2,190	103,065	12/31/24 to 1/1/31
Total	12/9/30	22	15,000	165	120,700	1/5/30 to 6/20/30

Table 3. - Lost-time accidents of the Old Dominion Co. and tho Phelps Dodge Corporation.

Properties	Fatal		Serious		Slight		Total	
	Number	Per 1000 shifts	Number	Per 1000 shifts	Number	Per 1000 shifts	Number	Per 1000 shifts
1924								
Copper Queen Branch	6	0.008	219	0.247	615	0.695	840	0.950
Moctezuma Copper Co.	9	.018	112	.224	738	1.478	859	1.720
Morenci Branch	1	.002	198	.450	267	.607	466	1.053
Old Dominion Co.	4	.014	68	.237	149	.519	221	.770
Four metal branches	20	0.009	597	0.283	1769	0.838	2386	1.130
1925								
Copper Queen Branch	2	0.002	166	0.184	315	0.349	483	0.535
Moctezuma Copper Co.	3	.005	51	.093	229	.418	283	.516
Morenci Branch	3	.007	133	.293	263	.578	399	.878
Old Dominion Co.	6	.019	81	.260	142	.455	229	.734
Four metal branches	14	0.006	431	0.195	949	0.428	1394	0.629
1926								
Copper Queen Branch	2	0.002	82	0.094	111	0.128	195	0.224
Moctezume Copper Co.	4	.007	26	.046	56	.100	86	.153
Morenci Branch	0	.000	49	.113	65	.151	114	.264
Old Dominion Co.	1	.003	31	.095	72	.222	104	.320
Four metal branches	7	0.003	138	0.086	304	0.139	499	0.228
1927								
Copper Queen Branch	6	0.007	81	0.093	45	0.052	132	0.152
Moctezuma Copper Co.	3	.005	16	.027	18	.030	37	.062
Morenci Branch	2	.005	18	.046	8	.020	28	.071
Old Dominion Co.	0	.000	34	.115	32	.109	66	.224
Total metal branches	11	0.005	149	0.069	103	0.048	263	0.122
Stag Canon Branch (coal)	5	.016	53	.175	109	.359	167	.550
Total all branches	16	0.007	202	0.082	212	0.086	430	0.175

Table 3. - Lost-time accidents of the Old Dominion Co. and the Phelps Dodge Corporation--Continued

1928

Properties	Fatal		Serious		Slight		Total	
	Number	Per 1000 shifts	Number	Per 1000 shifts	Number	Per 1000 shifts	Number	Per 1000 shifts
Copper Queen Branch	2	0.002	49	0.060	22	0.027	73	0.089
Moctezuma Copper Co.	6	.010	13	.021	8	.013	27	.044
Morenci Branch	5	.013	9	.024	2	.005	16	.042
Old Dominion Co.	1	.004	13	.051	3	.012	17	.067
Total metal branches	14	0.007	84	0.040	35	0.017	133	0.064
Stag Canon Branch(coal)	5	.020	44	.176	29	.117	78	.313
Total all branches	19	0.008	128	0.055	64	0.028	211	0.091

1929

Properties	Fatal Number	Fatal Per 1000 shifts	Serious Number	Serious Per 1000 shifts	Slight Number	Slight Per 1000 shifts	Total Number	Total Per 1000 shifts
Copper Queen Branch	3	0.003	41	0.040	5	0.005	49	0.048
Moctezuma Copper Co.	4	.006	16	.025	4	.007	24	.038
Morenci Branch	-	---	6	.013	-	---	6	.013
Old Dominion Co.	1	.004	10	.038	-	---	11	.042
Total metal branches	8	0.003	73	0.031	9	0.004	90	0.038
Stag Canon Branch(coal)	1	.005	22	.113	3	.015	26	.133
Total all branches	9	0.003	95	0.037	12	0.005	116	0.045

Four metal branches, per cent decrease in 1929 accident rate compared with year

	Fatal	Serious	Slight	Total
1928	57.2	22.2	76.5	40.6
1927	40.0	55.1	91.7	68.9
1926	-	64.0	97.1	83.3
1925	50.0	84.1	99.1	94.0
1924	66.6	83.2	99.5	96.3
1923	76.9	87.7	99.5	96.1

1930

Properties	Fatal Number	Fatal Per 1000 shifts	Serious Number	Serious Per 1000 shifts	Slight Number	Slight Per 1000 shifts	Total Number	Total Per 1000 shifts
Copper Queen Branch	2	0.003	24	0.036	3	0.004	29	0.043
Moctezuma Copper Co.	3	.006	14	.030	2	.004	19	.040
Morenci Branch	2	.005	6	.015	2	.005	10	.025
Old Dominion Co.	2	.008	10	.041	1	.004	13	.053
Total metal branches	9	0.005	54	0.030	8	0.005	71	0.040
Stag Canon Branch(coal)	1	.009	15	.140	2	.019	18	.168
Total all branches	10	0.005	69	0.037	10	0.005	89	0.047

CONCLUSION

The Old Dominion Co. has been able to create and maintain in the last few years an enviable safety record. This is all the more apparent when adverse working conditions are taken into consideration. Such an accomplishment has been made possible by the enthusiastic support given the safety department by every man from the manager down, the realization that safety starts at the top and must be treated as a major operating problem, an intelligent study of working conditions, and the adaptation of the principles of safety to those conditions.

APPENDIX

UNDERGROUND

Foremen and Bosses Safety Rules

The following instructions are issued to foremen and bosses of the Old Dominion Co.:

In case of a serious injury some one must be sent with the injured man to see that he has proper care until arrival at the hospital.

Blue cards are provided which must be filled out by the boss and issued to all injured men to take to the doctor for treatment.

All new men must be taken to the call-bell on the station of the level on which they are working and given full instruction in the use of the same for calling the cage, for the purpose of eliminating all chances of having any person ring the wrong bell.

Any boss who sees a man working under conditions which he considers dangerous, although he is working for another boss, must stop him immediately and have the other boss notified as quickly as possible.

Bosses who have untimbered drifts which they consider too dangerous to work without timbering, must take immediate action to secure the same, or place men elsewhere and notify their foreman.

Rigid inspection of drifts in the raw must be made to secure solid back and sides.

Timbered Drifts. All timbered drifts, either in ore or waste, which can be picked must be equipped with a top bridge high enough to take 4-inch spiling if necessary, and either plank or spiling must be put in as the shifter decides is necessary.

Timber in the level above working raises must be caught up before the raise is broken through. If the raise is coming through on the run of another foreman, the foreman in charge of the raise must notify him of the same and see to it himself that the work is done before breaking through.

Bosses must not allow one man to work alone in dangerous places.

Smoking shall be strictly prohibited on cages, moving man-trucks, timber stations, powder magazines, or while carrying or handling explosives.

See that men who are to be hoisted, line up back of turnsheet and take their regular turn without crowding. Bosses are instructed to discharge all violators.

Bosses must call to the attention of men fooling on the job with others, especially while riding on cages, the regulation prohibiting this practice, and see that it is strictly enforced.

The doorways on the skip compartments on stations where men get off and on must be kept clean.

At the time brass checks are given to new men going underground, the time clerk shall also issue to each man a pair of goggles with the printed instructions for wearing them.

It must be seen to that all men are provided with goggles and that they are being worn by them while engaged in breaking bowlders, starting holes, picking hard ground or moiling, barring down ground, blowing out holes, barring down chutes from manways, and all other work where there is danger of flying objects or dust.

Where men are not provided with goggles for use in their work, all eye injuries that might have been prevented by the wearing of goggles shall be classed against supervision, except in cases where inability to secure goggles has been previously reported.

Men working in drifts must use the light signals provided for that purpose to prevent being run into by motor trains; and before leaving, the signals must be removed.

Trolley-line boxes must be provided for men working around trolley lines, and bosses must insist on their use.

Motormen must leave the reverse lever on their motor in neutral, and remove trolley pole from wire whenever necessary for them to leave their motor.

Instruct motormen to slow down their trains when meeting men in drifts.

Discipline must be used with men caught coupling cars while the train is in motion, by giving them a short lay-off for the first offense, and discharging them for the second offense.

No person, including motorman and brakeman, shall get off or on motors, man-trucks or mine cars, while in motion.

Motormen either pulling or pushing trucks loaded with timbers must have a mine-car between the motor and timber truck to protect himself and helper against being caught with the timber.

All switches in main haulage drifts must be equipped with red lights and whistles, and the switch points must be left turned for the main haulage drifts at all times.

As soon as a missed hole is detected in a working place, all work must be stopped until the missed holes are blasted.

Men are strictly forbidden to enter any place after blasting in case of misfire, or for a period of 30 minutes from the time the fuse was lighted.

Hardwood skewers must be provided for boring holes in powder to insert caps.

Blasting switch boxes must be marked to correspond with the working place for which the box is used, and must have a tag attached bearing the words DANGER - DO NOT OPEN.

All timber being blasted must be done allowing safe time to return and make an inspection for fire before going off of shift.

Men blasting with electricity must be instructed to touch the ends of the lead wires together before connecting them to the wires from the holes. In case they get a spark when touching the wires they are to make a thorough inspection for the trouble and have it fixed before connecting the wires.

Drifts. All level development must be blasted with electricity from the start.

Raises. All level raises must be blasted with electricity from the butt caps. No deviation from this rule will be permitted except by first securing permission from the foreman.

Failure to secure blasting equipment will not make the use of fuse permissible, but must be reported to the foreman at once.

Before climbing up a working raise the air must be turned off, then turned on, and then turned off again with a short pause between each operation. Then get in communication with the men in the raise and if it is all right to climb up, turn the air on and go up. When down out of the raise and in the clear, signal to the men in the raise by turning the air off and on again.

All raise manways must be properly covered at the top while men are climbing up or down.

For hoisting and lowering timbers a chain must be attached to the rope and two hitches made around the timber.

The promiscuous dumping of carbide from lamps where the dust and fumes are offensive to other men in strictly forbidden. Bosses are requested to see that this rule is not being violated.

For erecting stages underground, the plank being used to nail against the posts must have two (2) nails at each end.

All manways not in good condition to travel through, also those that are in good condition but temporarily blocked by obstructions so that a person can not pass through, must be marked with a sign, CLOSED.

Temporary universal danger notices only, must be used as temporary danget signs for protecting men working in chutes, manways, etc. These notices must be properly filled in and dated, and removed immediately when no longer needed.

A brace, bit, and clevis must be supplied to the men, with instructions to use them wherever it is necessary to work undor timber being hoisted or lowered.

Dust must be dampened wherever possible, on machines and other places.

All intakes and the exposed working parts on the fan blowers must be kept guarded.

All loading places at chutos where men have to stand and load cars must be extended back so as to be not less than three (3) feet from the side of the cars.

See that care is exercised in piling timbers in main drifts and crosscuts, and have the timber kept in the timber stations wherever possible.

See that pipes in manways are put in so as not to interfere with men climbing on the ladders.

When putting in air or water lines, staples must be used for the small pipes, and hooks for the large pipes, to hold them in place.

Enough trolley-line boxes must be used to extend several feet each way from where repairmen are working.

The wearing of fuse for belts is strictly forbidden.

<u>Fire Prevention and Protection</u>. All abandoned workings, not needed for the operation of the mine, and in which there is a fire hazard, should be sealed with gunite stoppings.

All stations, rooms, and recesses constructed of wood, or other inflammable material for the installation of electrical equipment, must be thoroughly fireproofed.

Trolley wires must not touch caps of drift sets or projecting timbers.

All lamp sockets and fuse boxes must not be less than six inches from timbers.

Levels extending long distances from mine openings must be provided with permanent fire doors at intervals not exceeding 1000 feet.

The use of carbide lamps with swivel hooks is prohibted.

Careless men, especially those who have been previously warned, must be disciplined by a short lay-off for the first offense and discharge for the second offense.

Bosses on duty when mine is not working are in full charge and are responsible for existing conditions, and in case of emergencies, must take immediate action.

General Rules to be Observed for Mine Fires

(Amended March 15, 1927)

1. The first consideration must be the safety of all the men.

2. All air currents are to be maintained as nearly normal as possible until men are out of mine. Fans should not be reversed or stopped except on order from general mine foreman, superintendent, or manager.

3. All engineers must be thoroughly familiar with giving the fire signal.

4. Each shift boss must have a sufficient number of men chosen, instructed regarding the available exits and assigned to certain portions of his run, to enable him to get the men assembled as quickly as possible.

5. Powdermen and carmen must be thoroughly instructed regarding fire signals and should be kept informed at all times as to what men are chosen to make the assemblage and where they are working. Dated lists should be kept at the powder houses and revised at least once each month.

6. Any one discovering fire or dense smoke which can not be put out at once must immediately notify shift boss in charge of run, or if unable to locate him at once, then the powderman or assembly men must be notified, who will appoint some one to go to A shaft and ring the accident call, 7 BELLS, on the call bell, followed by level signal, for cage. When the cage arrives, he, himself, must go on top to the engine room and notify the engineer to give the fire signal, 3 - 3 - 3, on electric-light lines. He must then go to the time office and notify the clerks to call the mine-rescue station and mine officials.

7. The mine master mechanic, chief electrician and general mine foreman shall be notified immediately to send to all hoisting shafts a hoisting engineer and cage tender to get the men out of the mine, and an electrician to maintain the working of all the electrical equipment. All men assigned to these duties must remain there until further orders.

8. Shifters being notified of fire must have his powderman and assembly men notify all men to gather while he, himself, determines how to put out or control the fire. If there is any doubt as to the safety of the men he must proceed to get the men out at once.

9. When all the men are assembled they will proceed to the A shaft if possible, or if it is not possible to reach the A shaft without danger, they will take the best available exit.

10. The exit signs along drifts indicate way on all levels from A shaft to emergency exits.

11. All abandoned drifts must be marked by crossed lagging with a CLOSED sign.

12. Regardless of the exit used, each employee must call for his check at the time office at A shaft so that all men can be checked out.

13. It shall be the duty of all shift bosses to see that these rules are compiled with at all times.

A copy of the examination given crews trained in mine-rescue work follows:

QUESTIONNAIRE FOR DISTRICT BONUS MINE-RESCUE CREW,
GLOBE-MIAMI DISTRICT MINE-RESCUE AND FIRST-AID ASSOCIATION

Assembly of Apparatus

Have crew member who has tested his apparatus previously, throw apparatus on his back, couple it up and wear under oxygen in your presence for 5 minutes

The order for putting apparatus in operation is

(1) Throw apparatus on back (after goggles are on) - - ____________

(2) Put mouthpiece in mouth - - - - - - - - - - - - - - ____________

(3) Turn on "regular" valve - - - - - - - - - - - - - - ____________

(4) Put on nose clip - ____________

(5) Fill breathing bag 2/3rds full using by-pass valve ____________

(6) Tighten side straps - - - - - - - - - - - - - - - - - - - ____________

(7) Read gauge - ____________

Was this order observed? - - - - - - - - - - - - - ____________

Examiner will please note

(8) Are goggles right side up and is elastic tight enough? - ____________

(9) Is harness properly arranged? - - - - - - - - - - - - ____________

(10) Are all harness straps and buckles as they should be? ____________

Which ones wrong? ____________________

____

____

(11) Is mouthpiece "aimed properly"? - - - - - - - - - - - - - - ____________

(12) Is apparatus fully and properly equipped? - - - - - - - - ____________

(13) Nose clip? - ____________

(14) Mouth piece plug? - ____________

(15) Self Rescuer? - ____________

(16) If there are apparent leaks are they due to:

(17) Loose connections? - - - - - - - - - - - - - - - - - - ____________

(18) Lack of gaskets? - - - - - - - - - - - - - - - - - - - ____________

(19) Through nose? - ____________

Apparatus

(20) Make? - Draeger - - - - - - - - - - - - - - - - - - ____________

(21) Kind? - 2 hour - - - - - - - - - - - - - - - - - - - ____________

(22) Type? - Positive-pressure by-pass, fixed feed - - ____________

(23) Weight? - 43 lbs. - - - - - - - - - - - - - - - - - ____________

(24) Weight of oxygen cylinder? - 11 lbs. - - - - - - - - ____________

(25) Size of oxygen cylinder? - 2 liters - - - - - - - - ____________

(26) How much oxygen is in 2-hour cylinder when filled to 150 atmospheres pressure? - 300 liters - - - - - ____________

(27) How test for leaks? - (In water) capped and not capped (valve on and off) - - - - - - - - - - - - - - ____________

(28) How test for valve tightness? - Capped with valve open and valve closed - - - - - - - - - - - - - - - - - ____________

(29) Explain "blow off" on oxygen cylinder: - Located on valve head and should blow off at 3,000 lbs. pressure per sq. in. - ____________

(30) What is an atmosphere in pressure per sq. in.? 14.7 lbs. at sea level (about 13 lbs. here) - - - - - - - - ____________

(31) How much in volume is a liter? - 1 1/8 quarts (approx) ____________

(32) What is the pressure in pounds per sq. inch in full oxygen cylinder? - Approximately 1,950 lbs. - - - - ____________

(33) What part of apparatus is used to regulate the pressure while in operation? - Reducing valve - - - - - - - - -

(34) To what pressure is the bottle pressure reduced for operation & wearing? - 2.3 oz. per sq. in. - - - - -

(35) How much water gauge? - 10 centimeters or 4 inches -

(36) What part of apparatus regulates the flow of oxygen? - Reducing valve -

(37) How much oxygen should be admitted through the reducing valve to the breathing mixture each minute? - $2\frac{1}{4}$ liters

(38) What part of the apparatus produces the circulation? Injector -

(39) Where located? - In mixing chamber of reducing valve

(40) What rate of flow is provided per minute by this part? About 60 liters per minute - - - - - - - - - - - - - - -

(41) How measured? - By liter bag - - - - - - - - - - - - -

(42) What two kinds of regenerators are used? Cardoxide and Potash - - - - - - - - - - - - - - - - - - -

(43) Describe the cardoxide canister? - Copper refillable -

(44) Weight? - $5\frac{1}{2}$ lbs. - - - - - - - - - - - - - - - - - -

(45) How much cardoxide required to fill? - 4 lbs. - - -

(46) How long will charge serve? - 3 hours - - - - - - - -

(47) Intermittent or continuous? - Both - - - - - - - - -

(48) Temperature? - Cool - - - - - - - - - - - - - - - - - -

(49) Describe potash regenerator? - Not refillable; is a sealed tin canister - - - - - - - - - - - - - - - - - -

(50) Weight? - 4.2 lbs. - - - - - - - - - - - - - - - - - -

(51) Contents weight? - 2.2 lbs. - - - - - - - - - - - - -

(52) Number of screened shelves - 26 - - - - - - - - - - -

(53) Shelf area? - $9\frac{1}{2}$ sq. feet - - - - - - - - - - - - - - -

(54) How long will fresh canister serve? - 2 hours - - -

(55) Continuous or intermittent? - Continuous only - - - ____________

(56) Temperature while using? - Gets hot - - - - - - - - ____________

(57) What provision is made for cooling the air that becomes heated in the regenerator? - Air cooler - - - - - - - ____________

Describe the breathing bag

(58) Direction of flow of air in or out of each hole? - - - ____________

(a) Fresh air enters lower opening under right side and leaves upper right front - - - - - - - - - - - - - ____________

(b) Exhaled air enters upper front left side and goes out left lower underside - - - - - - - - - - - - - ____________

(59) Is there an opening between the two sides of breathing bag? - Yes - ____________

(60) Why? - To furnish air to the wearer when his requirements are in excess of amount available in fresh air side of bag - ____________

(61) Is there a release valve? - Yes, but it has been blocked as it releases too easy - - - - - - - - - - - ____________

(62) How do you release when apparatus is too full? Let air out around mouthpiece while squeezing bag - - ____________

Describe the by-pass

(63) Location? - Angle valve on oxygen cylinder - - - - - ____________

(64) Action? - Admits oxygen through by-pass tube to mixing chamber - ____________

(65) When used? - For filling apparatus to test for leaks ____________

(66) For filling apparatus to start wearing - - - - - - - ____________

(67) For filling apparatus after scrubbing apparatus - - - ____________

(68) For filling apparatus when temporarily short on air - ____________

(69) For filling apparatus to make apparatus usuable when reducing valve has failed or apparatus is rendered out of order while wearing - - - - - - - - - - - - - - ____________

(70) Describe scrubbing of apparatus or nitrogen accumulation - ____________

If at the end of one hour the apparatus is full of air, the breathing bag should be squeezed empty and the apparatus refilled by using the by-pass - - - - - - - ____________

(71) Why is scrubbing necessary? - The oxygen used contains as high as 2% nitrogen. This nitrogen can accumulate until at the end of one hour the mixture of air in apparatus could have an excess of nitrogen, which is occupying space needed for oxygen - - - - - - ____________

(72) Describe the dial readings on pressure gauge? Atmosphere pressure and minutes per hour - - - - - - ____________

(73) 50 atmospheres is good for how many minutes? - 40 min. ____________

(74) 100 " " " " " " " - 80 " ____________

(75) 150 " " " " " " " -120 " ____________

Describe the mouthpiece

(76) Direction of intake and outgo? - Right to left - - - ____________

(77) Valves are for what purpose? To control the direction - - - - - - - - - - - - - - - ____________

(78) How can one test these valves to determine if flow is in proper direction while wearing? - By collapsing first one respiration tube and then the other - - - - ____________

What is the direction of flow of air in

(79) Right side of breathing bag? - Up - - - - - - - - - - ____________

(80) Left side of breathing bag? - Down - - - - - - - - - - ____________

(81) Cooler? - Down - ____________

(82) Regenerator? - Up - - - - - - - - - - - - - - - - - - ____________

(83) In the long stiff circulation tube? - From bottom of cooler to breathing bag - - - - - - - - - - - - - - - - - ____________

(84) In short stiff circulation tube? From breathing bag to mixing chamber - ____________

Is the composition of air mixture "good" or "bad" or all oxygen in the

(85) Cooler? - Good (except for cooling) - - - - - - - - - ____________

(86) Mixing chamber? - Good and bad - - - - - - - - - - ______________

(87) Right side of breathing bag? - Good - - - - - - - - - ______________

(88) Left side of breathing bag? - Bad - - - - - - - - - - ______________

(89) Does "good" and "bad" or just "bad" air enter the regenerator? - Both - - - - - - - - - - - - - - - - - ______________

Explain 4 tests for

(90) Cardoxide refillable regenerator? - Freshness of charge which you made yourself or by cotton and adhesive tape already on cartridge opening - - - - - ______________

(91) For leaks? - By blowing into it - - - - - - - - - - ______________

(92) For stoppage? - By blowing through it - - - - - - - ______________

(93) For bent or damaged ends? - - - - - - - - - - - - - - ______________

Explain 5 tests for

(94) Potash regenerator? - Freshness by seals and shaking to note if contents are granular - - - - - - - - - - ______________

(95) For tightness by blowing into - - - - - - - - - - - ______________

(96) For stoppage by blowing through - - - - - - - - - - - ______________

(97) For bent or damaged ends - - - - - - - - - - - - - - ______________

(98) Screens too close to ends - - - - - - - - - - - - - - ______________

(99) How do you test for low pressure leaks in apparatus? Block mouthpiece with one hand and fill apparatus, using by-pass valve. If leaks are present the bag will wilt noticeably in 15 seconds - - - - - - - - - ______________

(100) Why should you listen for "hum"? - A "good" hum will denote a proper amount of air and oxygen in circulation - ______________

(101) How do you test for high pressure leaks and where? With spark on brass parts only. Around oxygen cylinder valves, wrench connection, around reducing valve union, around to pressure gauge, on reducing valve blow off and around pressure gauge - - - - - - - - - ______________

(102) What are the cleaning and sterilizing requirements of apparatus and for what parts? - Rinse - soap with brush, rinse in fresh water, then in lysol solution, corrosive sublimate solution and then in potassium permanganate solution and hot water, rinse (live steam if available)
Mouthpiece, respiration tubes, breathing bag - - - - ______________

(103) While wearing the apparatus, 300 liters of oxygen is added to the contents of the apparatus from the oxygen cylinder during the two hours - - - - - - - -
An equal volume is inhaled and exhaled at each breath. What becomes of this extra 300 liters in volume? -
The oxygen enters the lungs as such and part is changed into carbon dioxide which is exhaled and absorbed by the contents of the regenerator - - - - - - - - - - - - ______________

(104) How many men shall compose a crew? - Five - - - - ______________

(105) Is it permissible for one man to return to fresh air alone? - No - ______________

Demonstrate how to use

(106) Water gauge on apparatus - - - - - - - - - - - - - - - - - ______________

(107) Liter bag on apparatus - - - - - - - - - - - - - - - - - - - ______________

(108) How much of the apparatus has a negative pressure while in operation? - From left side of mouth piece through left respiration tube, left side of breathing bag and through short stiff circulation tube to vacuum chamber - ______________

Half-hour Apparatus

(109) For what purpose is half-hour apparatus used?
For exploration or errand work or to take to men to use who have to be brought through smoke, or is to be worn at time of Mine Fire by those traveling in the mine who might encounter smoke - - - ______________

(110) Describe apparatus

(111) Is good for half-hour only - - - - - - - - - - - - - - - ______________
Is a by-pass type apparatus - - - - - - - - - - - - - - ______________

(112) How worn and used? - Have crew member put it in position to wear and show how to operate and read gauge - ______________

(113) Where are they stored at applicant's plant? - - - - ______________

(114) Where are the two on the rescue truck stored?
On right front of top deck - - - - - - - - - - - - - ______________

(115) How full is the oxygen cylinder filled?
150 atmospheres - ______________

(116) Describe regenerators? - Cardoxide, Potash - - - - ______________

Oxygen Pumps

(117) Are the thread connections a standard or metric thread? - Metric - - - - - - - - - - - - - - - - - - - ______________

(118) How can you connect the large oxygen cylinder which has standards threads to the pump, that has only metric threads? - By using an "adaptor" - - - - - - ______________
(Have crew member pick one out from the articles on desk) - ______________

(119) Why use three large cylinders for the pump?
To enable the use of more of the oxygen from each large cylinder, by using from the low cylindeґs in filling "low" bottles - - - - - - - - - - - - - - - - - - ______________

(120) Why is only one large cylinder opened at a time?
Because the large cylinders would all equalize in pressure if all three were opened at once - - - - - ______________

(121) How can you tell the pressure in the large cylinders while attached to pump? - By opening the cylinder valve and reading upper gage - - - - - - - - - - - - - ______________

(122) How can you tell how full the apparatus bottle is while filling? - By reading lower gage with apparatus bottle valve open - - - - - - - - - - - - - - ______________

(123) What is used for lubricant? - $\frac{1}{4}$ glycerine and $\frac{3}{4}$ water - ______________

(124) Why not use oil? - Compressed oxygen with oil will cause an explosion - - - - - - - - - - - - - - - - - - ______________

(125) Crew member will be taken to pump and indicate how he would fill an empty apparatus bottle from three cylinders with pressure of 50 Atms. 90 Atms. 120 Atms. ______________

Gas Masks

(126) Indicate how worn

(127) How put into service - Test for leaks by placing hand over intake opening at bottom - - - - - - - - - ______

(128) Note gage on air indicator - - - - - - - - - - - - - - ______

(129) Remove stopper from bottom of canister - - - - - - ______

(130) What gases will it protect you from?
Carbon monoxide
Ammonia
Illuminating Gas
Smoke
Timber gases - ______

(131) Weight? - $5\frac{3}{4}$ lbs. - - - - - - - - - - - - - - - - - ______

(132) How long will it serve? - For two hours or more, depending on how much gas it has to absorb - - - - ______

(133) Does the use have to be continuous? - No, it can be worn intermittently for one year, provided it is kept stopped when not in use or is not used in excessively moist air - - - - - - - - - - - - - - ______

(134) How can you tell when the canister is beginning to fail? - By its getting noticeably warm - - - - - ______

(135) Should it be warm when the oxygen content of air is very low? - No - - - - - - - - - - - - - - - - ______

(136) How low does the oxygen content of air in a metal mine have to be for danger in wearing gas masks? - Oxygen too low for a candle or carbide lamp to burn freely - - - - - - - - - - - - - - ______

(137) Should crews wearing gas masks be equipped with flashlamps only? - No, should have flashlamps and carbide lamps or lighted candles - - - - - - - - ______

(138) Where stored on rescue truck? - Six with extra canister are stored in long cupboard at top of left side of truck at rear - - - - - - - - - - - - ______

Self-Rescuers (Carbon Monoxide)

(139 How carried on apparatus? - Snapped to right side at waistline - ______

(140) How opened? - Tear off the flat side that snap is fastened to - ______

(141) What gases will it protect the wearer from?
Carbon monoxide
Smoke and timber gas - - - - - - - - - - - - - - ______

(142) Weight in container? - 28 oz. - - - - - - - - - - ______

(143) Weight as used? - 13 oz. - - - - - - - - - - - - - - ______

(144) How long is it good for? - For half-hour or more, depending on how much gas it has to absorb - - - ______

(145) Can it be used intermittently or must it be used continuously? - Intermittently for days if kept in dry place - ______

(146) Does it furnish wearer any oxygen? - No - - - - ______

(147) Should it be worn where the oxygen content of air is very low? - No - - - - - - - - - - - - - - - - - - - ______

(148) How can you determine in a metal mine, following a mine fire, if there is sufficient oxygen to risk wearing this self-rescuer? - By candle or carbide lamp. If there is sufficient oxygen for a carbide lamp or candle to burn freely there is sufficient oxygen to support life - - - - - - - - - - - - - - - - ______

Fire Extinguishers

Soda Acid

(149) How used? - Turn upside down and shake - - - - - ______

(150) How recharged?

(a) Bottle in holder at top filled up to mark with sulphuric acid.
(b) Lead gravity stopper placed in bottle.
(c) Baking soda dissolved in water in tank up to mark on inside - - - - - - - - - - - ______

(151) How many soda-acid extinguishers on rescue truck? Six - ______

(152) Where? - At front end of gangway - - - - - - - - - ______

Pyrene Fire Extinguishers

(153) How used? - Unlatch and pump - - - - - - - - - - ______

(154) How refilled? - Unscrew cap and fill with liquid pyrene -
(Three 1-gallon cans of pyrene are kept on rescue truck at front end of gangway)

Mine Gases

(155) What gases following a metal mine fire may usually be expected other than oxygen and nitrogen - - - - - -
Carbon dioxide
Carbon monoxide
Sulphur dioxide - - - - - - - - - - - - - - - - -

(156) If candle will not burn in the air, what condition of air is indicated? - Oxygen content is less than $16\frac{1}{2}\%$, or carbon dioxide is present and air is not fit to remain in without apparatus - - - - - - - - - -

(157) How can carbon monoxide be tested for (two ways)? By using canary birds. If canary shows marked distress in three minutes it is unsafe to remain without apparatus
By using carbon monoxide detector, if any carbon monoxide is detected it is unsafe to remain without apparatus -

(158) How much carbon monoxide will cause distress to a person? - 0.05%, slight poisoning in one hour; 0.12%, 30 minutes bad, one hour stagger, two hours violently and dangerously sick; 0.20%, very dangerous -

(159) How low does oxygen content of air need be to cause distress? - 13% distress, 7% fatal - - - - - -

(160) How would you take a gas sample? (two methods)
(a) Emptying a bottle of water in place air is to be sampled
(b) Using vacuum bottle (demonstrate) - - - - -

(161) Precautions?
(a) Get all of water out of bottle
(b) Stopper tightly
(c) Label carefully - - - - - - - - - - - - - -

(162) Temperatures
Have crew member read thermometer held in heat register -

(163) How is velocity of air determined? - By anemometer

(164) How is it read? - In feet per minute - - - - - ______

(165) How is volume calculated? - By multiplying area at place reading was taken by the velocity per minute - ______
(Have crew member read anemometer)

(166) If water gage was required, how could it be determined? - By connecting one end of U tube to hole in stopping and reading difference in height of water in the U tube - - - - - - - - - - - - - - ______
(Have crew member read a water gage)

(167) How much air would an entombed man require each 24 hours? - If he remains at rest 400 cubic feet (1 cubic foot = 28.32 liters) - - - - - - - - - - ______

(168) If he works some? - Laboring some he would require double that amount or 800 cu. ft. - - - - - ______

Walking at the rate of 2 miles per hour a man needs about double the amount of air required while at rest
Walking at the rate of 4 miles per hour a man needs about 4 times the amount of air required while at rest.

(169) Demonstrate how to use monoxide detector - - - - ______

(170) How many times should bulb be squeezed? - Ten - ______

(171) Suppose tube showed very blue at first squeeze? (High CO and unsafe to remain without apparatus) ______

Miscellaneous Accessory Equipment

(172) Telephone

How many field telephones have we? - Three - - ______

How much wire on reels? - 3,000 feet - - - - - ______

(173) How operated to call and talk? - Ring by turning crank and push button in on hand piece to talk and hear (demonstrate) - - - - - - - - - - - - - - - - ______

(174) Can one phone be used while it is being moved? Yes, if attached to "contact" reel one phone can be used - ______

Pipe connections

(175) Cellar nozzles.

How many do we have? - Two - - - - - - - - - - - -
How many openings on each? - 9 and 4 - - - - - - - ______________

(176) What size pipe do they fit? - 1 and $2\frac{1}{2}$ inch - - -
Can size of pipes serving them vary? Yes, by using bushings or reducers - - - - - - - - - - - - - - - - - - ______________

Geophone

(177) For what is it used? - To hear sounds through the ground - ______________

(178) Can direction of sound be determined by use of geophone? - Yes - ______________

(179) Where stored on truck? - In long cupboard at top of body at rear on right side - - - - - - - - - - - ______________

(180) What is the pyrotannic detector used for?
To determine the per cent of carbon monoxide the blood has absorbed - - - - - - - - - - - - - - - - - - ______________

(181) Can the blood of a dead person be tested? - Yes ______________

(182) What electric current will our two fans operate on?
One, 110V, 220V, 400V. A.C.
Other 220V. D.C. - - - - - - - - - - - - - - - - - - - ______________

(183) What gages of track will they run over?
18, 24, 30, and 36 inch gage - - - - - - - - - - - - ______________

(184) What track gage will serve the crew member's mine? ______________

(185) What fan will serve the crew member's mine? - - - - ______________

Old Dominion. The D.C. everywhere (460 r.p.m.) and the A.C. (330 r.p.m.) on 18 station and the 18 W. and 2012 winzes.

Arizona Commercial M.Co. 220 A.C. (335 r.p.m.)

Iron Cap C.Co. (110 A.C. - 350 r.p.m.) Lighting circuit
(440 A.C. - 490 r.p.m.) Shaft and station
(110 A.C. - 520 r.p.m.) Surface

Miami Copper Co. 220 D.C. - 477 r.p.m.

Inspiration C.C.Co 440 A.C. - 510 r.p.m.

(186) Have you read a copy of the instructions in case of fire at your mine? - - - - - - - - - - - - - - - - ____________

If crew member has not, furnish him with copy to read. After reading credit him with "Yes". (The answer to this must be "Yes."

Precautions around Apparatus

(1) Be careful of leaks in apparatus.

(2) Be sure of full cylinder of oxygen.

(3) Be sure of freshness of regenerator charge.

(4) Keep all open lights (carbide lamps, etc.) away from apparatus while assembling and especially while wearing.

(5) Never let the members of a 5-man crew become separated.

(6) Always use life line even though air is clear when entering.

(7) Use "blind" man's sticks when entering heavy smoke.

(8) Do not travel in places except where agreed upon when entering.

(9) Never climb up or down raises without orders and then all possible precautions will be taken, such as posting a reserve crew at the entrance to the raise and another reserve crew at fresh air base.

(10) When the by-pass is used several times in a few minutes by one or more members of the party, it should be investigated, if nothing is found wrong the pace should be reduced.

(11) No crew member should be alone on a cage deck.

(187) How many of the above suggested precautions can the crew member relate? - - - - - - - - - - - - - - - - ____________

I. C. 6547 August, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XXVIII

BY

FREDERICK W. LEE

I.C.6547
August, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]

No. 28

Compiled by Frederick W. Lee[2]

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1. GRAVITATIONAL METHODS

(315) PERFECTIONNEMENTS A L'INSTRUMENT TRANSPORTABLE POUR LA MESURE RAPIDE DE LA GRAVITÉ

(IMPROVEMENTS MADE ON A PORTABLE INSTRUMENT FOR QUICK MEASUREMENT OF GRAVITY)

By F. Holweck and P. Lejay

Comptes rendus de l'Académie des Sciences, Paris, vol. 192, No. 18, 1931, pp. 1116-1119

The principles and construction of an instrument for quick measurement of gravity were explained by the authors in a previous article published in the Comptes rendus de l'Académie des Sciences, vol. 190, No. 24, 1930 (see Geophys. Abs. No. 21, p. 2).

In this article the improvements made on the instrument are described and illustrated by a schematical drawing. The method of observation has also been changed considerably by abolishing the recording and replacing it by direct stroboscopic observation. Fifteen minutes are required for making one observation.

Measurements of gravity were carried out near Dijon in order to test the instrument, and it was established that the new model was ten times more accurate than the old one. The instrument was not affected by being transported in an automobile for more than 2,000 kilometers without any special protection.—W. Ayvazoglou.

(316) GRAVITY GRADIENT AND CURVATURE

By Edith A. Nelson

Philosophical Magazine No. 10, Sept., 1930, pp. 513-516

A survey with the Eötvös balance shows that in certain instances the curvature of the level surface changes sign when the gravity gradient is a maximum, while in other cases this does not hold. Investigation shows that there is no general relation between the quantities, and that this phenomenon is evidence of stratification in that region of the earth's crust, the edge of the stratum being situated below the point of the maximum gradient.--Abstract from Science Abstracts, vol. 34, No. 398, 1931, p. 96.

(317) DAS GEOID

(THE GEOID)

By F. Ackerl

Gerlands Beiträge zur Geophysik, Leipzig, vol. 29, No. 3/4, 1931, pp. 273-335.

In this article Ackerl considers the question of the mathematical figure of the earth according to theories advanced by Hopfner and developed by Bruns, as he thinks it desirable to give numerical calculations of the theoretical results. The article contains the following items:

A. Theoretical argumentations.
B. Observation material.
C. Sources of reference.
D. Remarks to the sources of reference.
E. Lines of equal gravity.
F. Special maps of G lines.
G. World map of lines of equal gravity.
H. Determination of the apparent level spheroid.
I. The world map with the lines of equal apparent gravity anomalies.
K. Special maps of apparent gravity anomalies.
L. The presumable value of the undulations of the geoid.
M. Conclusive remarks.

A translation of the author's abstract reads as follows:

F. Hopfner proposed a hypothesis-free method of determining the shape of the geoids. This method requires values of gravity which are derived from the gravity observed according to well-founded viewpoints established in physics. All values of gravity published till July, 1930 (as far as they were known by the author), are reduced by A. Prey's method to correspond with the sea level of Vienna, which is taken as the starting point.

By using these gravity values a world map is drawn in which the course of lines of equal gravity is shown. The constants of the apparent level spheroid are computed by a preliminary application of the summary method. The calculated anomalies are used for drawing a world map in which the lines of equal apparent gravity anomalies are represented. From the amount and distribution of these anomalies a conclusion as to the amount and distribution of undulations of the geoid with regard to the true level spheroid can be drawn.--W. Ayvazoglou.

(318) DAS SCHWEREPROFIL DER TAUERNBAHN

(THE GRAVITY PROFILE OF THE TAUERN RAILROAD)

By Robert Schwinner

Gerlands Beiträge zur Geophysik, Leipzig, vol. 29, No. 3/4, 1931, pp. 352-416.

The investigation is based on the determination of gravity made in 1910-1912 along the Tauern railroad and the parallel profile Rauris-Sonnblick-Ober-Drauburg, and especially on the determinations of gravity anomalies $G_0-\gamma_0$ -- that is, on the observed gravity, reduced to sea level according to Bouguer's method, less the normal gravity γ_0, the latter being assumed according to Helmert's formula of 1901. In the first place the question of the density of rocks, which is fundamental to all gravimetric problems, is discussed in general and in particular with respect to the rocks found in the Tauern profile; improvements made on Bouguer's reduction formula at several stations resulted in considerable changes of the final values of anomalies to be applied.

The problem was to find, by investigation based on geological and geophysical premises, a disposition of the subterranean disturbing masses, such that the gravity anomalies $G_0-\gamma_0$ resulting from the reduction of the measurements might be explained by their attraction. Owing to the fact that these "measured anomalies" in this case, as well as in the Alps in general, are all negative only a negative disturbing mass was to be considered--that is, rocks which had a lighter specific weight than the surroundings at the same level.

Local disturbances were caused chiefly by the Tauern granite (density of the disturbing mass = δ = -0.15) and in much less degree by the calcareous phyllite (δ = -0.05) of which only its accumulation on the northern slope was taken into consideration. For the granite resulted as the most probable form, that of a horizontal plate or a flat lens, about 8 to 9 kilometers thick, at the northern border of which a vertical root or a dyke, of at least 5 kilometers in magnitude, is placed. The supposition of some geologists that the granitic masses of the Tauern are dipping at their southern border in the forms of large roots or dykes does not agree with the present determinations of gravity.

The regional negative gravity in the Alpine region is represented (according to Airy) as the effect of a "mass deficiency" caused by the fact that under the Alps the acid rocks of the outer crust (sial) are sinking, owing to their weight, lower into the basic substratum (sima) than the neighboring areas; the form can be simplified and represented as a rectangular Graben in the sima filled with sial (δ = 2.8 - 3.1 = -0.3) descending at the southern border from 30 to 40 kilometers and at the northern border rising in two steps from 40 to 35 and from 35 to 30 kilometers to the normal depth of the sial-sima bordering plain which, according to seismic data, is supposed to be 30 kilometers only. These three steps correspond to the trend line of

the principal faults of the Alpine structure; the line of the Drau, of the Salzach and the border of the Alps; it is supposed that the two former are dipping northward and the latter southward. From this assumption a disturbance of gravity is computed which is sufficiently in accordance with the anomalies determined, especially because for the smaller variations (Sonnblick northern slope, Bischofshofen-Hochkönig, Hallein-Berchtesgaden) the local causes of disturbance can be proved as resulting from geological conditions.

Based on these assumptions on the disposition of masses the hydrostatic uplift was estimated, corresponding to the conception resulting from Airy's hypothesis that the floating of sial is similar to the floating of pack ice. Prevailing of weight was established in the region of the Drau; prevailing of uplift was found at the northern border and far beyond the Solzach and prevailing of weight again in the country of Berchtesgaden. This corresponds to the fact that the region of the Drau and the Carinthian Lakes, as well as the region of Berchtesgaden, are proved to be sinking during centuries, while the Tauern and the region to the north, up to the Hochkönig, are being lifted up.

Geologically, the data of the gravity profile and the supposed masses seem to prove that in this region vertical movements play the main part (in some way according to K. Lehmann's "Pingen-Theory") and speak against the horizontal dislocation, the latter idea being recently emphasized too much.

In conclusion some directive lines for the continuation of the gravity measurements in the Tauern are given.--Author's abstract translated by W. Ayvazoglou.

(319) EIN BEISPIEL ZUR ENTWICKLUNG DES RAUMPOTENTIALS NACH KUGELFUNKTIONEN

(AN EXAMPLE FOR THE RESOLUTION OF THE SPACE-POTENTIAL ACCORDING TO SPHERICAL FUNCTIONS

By K. Jung

Gerlands Beiträge zur Geophysik, Leipzig, vol. 29, No. 3/4, 1931, pp. 346-351.

Using a simple and evident example (a homogeneous oblate ellipsoid of revolution) the author shows that the resolution of the spherical functions of the external space potential can not be applied inside of masses, also not near the surface and even not if the series of the external space potential converges in one part of the mass volume.--Author's abstract translated by W. Ayvazoglou.

2. MAGNETIC METHODS

(320) REPORT ON THE MAGNETIC SURVEYING IN THE YEARS 1924-1925 OF THE MAGNETITE SCHISTS IN THE REGION OF THE KOLA-FIORD AND ON THEIR POSSIBLE ECONOMIC IMPORTANCE (IN RUSSIAN)

By A. S. Purtov

Bulletin of the Geological and Prospecting Service, Moscow, U.S.S.R., vol. 49, No. 10, 1930, pp. 45-72.

The article consists of two parts. In the first the magnetometric survey carried out in 1924-1925 is described; in the second the author gives an economic estimation of the reserves of magnetite deposits based on the data obtained from this survey.

An area of about 110 square kilometers was covered by the survey, 9,500 observations being made. Thalen-Tiberg's magnetometer was used.

The main results obtained by the survey are summed up as follows:

1. The width of the ore-bearing complex was determined to be considerable.
2. The distribution of the separate orebodies in the complex was found to be scattered.
3. A series of new strong anomalies was discovered.
4. The strongest anomalies were found to be in the northern part of the complex.
5. Finally, indications concerning the relations between the different ore types and the intensity of the anomalies were established.

The possible total reserves of all the largest deposits calculated on the basis of the data obtained from the magnetic survey are considered to be about 24 millions tons.--W. Ayvazoglou.

(321) RECONNAISSANCE MAGNÉTIQUE EN SYRIE

(MAGNETIC INVESTIGATION IN SYRIA)

By Jean Chevrier

Comptes Rendus de l'Académie des Sciences, Paris, vol. 192, No. 16, 1931, pp. 977-978.

The purpose of this investigation was to determine the magnetic elements (declination, inclination, intensity of the horizontal component) in the desert regions. The observations were carried out by a theodolite compass (Brunner-Chasselon model No. 3) and an inclination needle (Chasselon model No. 1).

The error in declination may be considered to be about 2 minutes for the most unfavorable conditions. An error of the same order may be expected for inclination. The error in determining the horizontal component has proved to be very small.

A comparison of values obtained during this investigation in Karyatein and in Palmyre with declinations observed by the Carnegie expedition in 1911 in the same places, shows a secular variation of 6.5 toward the east.

The results of observations are given in a table.--W. Ayvazoglou.

(322) GENERAL PRINCIPLES FOR DETERMINING THE EXTENSION AND DEPTH OF AN OREBODY HAVING THE SHAPE OF AN OBLATE ELLIPSOID OF REVOLUTION MAGNETIZED PERPENDICULAR TO THE AXIS OF REVOLUTION (IN RUSSIAN)

By N. Boulgakov and A. Jouravsky

Bulletin of the Geological and Prospecting Service, Leningrad, U.S.S.R., vol. 49, No. 5, 1930, pp. 61-73.

In this paper the authors study the magnetic field of an orebody having the shape of an oblate ellipsoid of revolution and being magnetized in the direction perpendicular to the axis of revolution. Tables are computed with the aid of which the components X, Y, and Z for different points of the field may be easily determined.--W. Ayvazoglou.

3. SEISMIC METHODS

(323) SHOCK DISPERSION

By K. Sezawa and G. Nishimura

Bulletin of the Earthquake Research Institute, Tokyo, vol. 8, No. 3, 1930, pp. 321-337.

That part of transmitted waves dispersed by an echoing medium is not of regular form, while the oscillatory part of ordinary dispersed waves is quite regular and has gradually varying wave length. In echoed dispersion the leading part is distinct and has large amplitude. In ordinary dispersion both leading and trailing parts have their own velocities of propagation. A portion of the vibrations of echoing dispersion is of a gradually decaying type, whilst in ordinary dispersion this is not in general possible.--Author's abstract.

(324) SEISMOGRAPHEN IM DIENSTE DER GLETSCHERFORSCHUNG

(SEISMOGRAPHS IN THE SERVICE OF GLACIER INVESTIGATION)

By Hans Mothes

Forschungen and Fortschritte, Berlin, vol. 6, No. 28, 1930, pp. 363-365.

The importance of a knowledge of the thickness of glacier ice in geophysical prospecting, as well as in climatology and the study of isostatic equilibrium of the earth crust, was the cause of investigations of glaciers by means of geophysical methods which have been undertaken since 1925.

In this article the author gives a description of his first practical work on glacier investigations carried out in August, 1926, in Hintereisferner (Ötztaler Alps). This place was chosen because it had already been extensively studied by H. Hess.

Wiechert's field vertical seismometer was used. Seismograms as well as travel-time curves are given in figures. Of special interest are curves showing hyperbolic form, as theoretically they represent waves reflected in the underground during their path from the point of explosion to the point of observation. Greater distances of these curves from the abcissa indicate greater thickness of the ice stratum and vice versa. The path of these waves and therefore the depth of the reflection underground could be calculated from the travel time taken from the seismograms; of course their velocity was to be known. This velocity has been established to be 3,600 meters per second.

The correctness of data obtained by seismic method of investigation was proved by borings carried out by Hess and others, as well as by additional investigations made by the author and published in Zeitschrift für Geophysik, vol. 6, Nos. 3-4, 1929 (see Geophys. Abs. No. 9, p. 13). Similar investigations were made by the author in August, 1929, on the Pasterze Glacier (Eastern Alps) and were published in Zeitschrift für Geophysik, vol. 6, No. 8, 1930 (see Geophys. Abs. No. 21, p. 7).--W. Ayvazoglou.

(325) ÜBER DIE HERDTIEFE DER ERDBEBEN

(ON THE DEPTH OF EARTHQUAKES' FOCI)

By Gerhard Krumbach

Forschungen und Fortschritte, Berlin, vol. 6, No. 32, 1930, pp. 419-420.

Two methods for the determination of the focus of an earthquake are pointed out by the author:

1. The macroseismic method -- in which the propagation of the intensity of a quake on the surface of the earth as resulting from observed shocks and destructions in the area of the quake is studied (determination of isoseismal lines).

2. The microseismic method -- in which the records of seismic instruments are used and the periods of time of propagation of definite forms of seismic waves are studied in dependency from the distance of the focus; that is, travel-time curves are drawn (determination of isochronous lines). Both methods are discussed. Krumbach concludes that although a series of important results on investigations concerning the depth of the foci of earthquakes have already been obtained, this work must be continued in order to get valuable information on the structural changes inside of the earth crust.--W. Ayvazoglou.

(326) ON DEEP-FOCUS EARTHQUAKES

By R. Stoneley

Gerlands Beiträge zur Geophysik, Leipzig, vol. 29, No. 3/4, 1931, pp. 417-435.

Professor Turner has found that the foci of earthquakes may occasionally be situated as low as 0.09 of the earth's radius below normal; that is, they may occur 580 kilometers below the surface.

Allowance for the known errors of the Zöppritz-Turner tables would probably reduce these depths considerably.

According to a general reciprocal theorem in dynamics, since the amplitudes of surface waves fall off rapidly at great depths, the surface waves of deep-focus earthquakes should be small or insensible. Yet the International Seismological Summary gives L and M readings even for the deepest foci contemplated.

It is here shown that in some of the deepest focus earthquakes, the recorded L and M are scarce and often nonexistent at great distances. The recorded observations mostly refer to S, SS, SSS, etc., or else to Gutenberg's early long wave G, for which the almost complete extinction is not to be expected, as the period is very long.

The actual records of these very deep-seated shocks show a very conspicuous P, a large S, SS, and further disturbances, and no L or M. The amplitudes of the general disturbance at the calculated positions of L and M are smaller, often much smaller, than that of P.

Finally, a tribute must be paid to the pioneer work of the late Professor Turner. His deep-focus earthquakes, even if not of such great depth as he thought, have at any rate a focus in some cases far below normal.--Author's abstract.

(327) SEISMOMETRIC STUDY OF THE RECENT DESTRUCTIVE NORTH IDU EARTHQUAKE

By A. Imamura

Bulletin of the Earthquake Research Institute, Tokyo, vol. 9, No. 1, 1931, pp. 36-49.

In this article Imamura describes seismometric studies carried out in connection with the earthquake in North Idu which occurred on November 26, 1930. The results of the investigation are summed up as follows:

1. Of various earthquake phenomena, the outstanding one is the relation between the distributions of faults and seismic intensities. Of the number of faults that appeared, four were found especially noteworthy; a description of them is given.

2. Another characteristic feature was observed in the seismograms obtained at Tokyo. The result of the study of the diagrams is given in a figure. It shows the time of the commencement and of the duration of the tremors, and the period and amplitude of the S phase. Further analysis shows that the earthquake consisted of waves which may be regarded as having emerged from four different sources.

3. Another interesting feature is the presence of vibrations of a period of some 10 seconds from the very beginning to the last, and attaining after 30 seconds from the commencement a double amplitude of so much as 14.7 centimeters--largest amplitude so far perfectly recorded with instruments in Japan. These long waves were propagated to Tokyo first as waves of dilatational type and then as those of distortional type; they may be regarded as caused by the bodily movement of the crustal block constituting the seismic region.

The block movement thus seems to have started somewhere near the middle point of the fault system and passed on one hand quietly over the meridional fault toward the north and on the other hand toward the south, sharp destructive shocks having emerged from both of the three oblique faults.--W. Ayvazoglou.

4. ELECTRICAL METHODS

(328) SALT DOME STUDIES BY GEOELECTRICAL METHODS

By Karl Sundberg

The Petroleum Times, London, vol. 25, No. 638, 1931, p. 507.

After a brief explanation of Swedish electromagnetic method for structural studies, Sundberg concludes that geoelectric methods were successful in the detailed determinations of the flank conditions of salt domes, and especially in locating faults against which oil might accumulate, discovered in the Gulf Coast region by geophysical, chiefly seismic, methods.

Two figures representing electromagnetic investigations across salt-dome structures are given.--W. Ayvazoglou.

5. RADIOACTIVE METHODS

(329) ULTRA-PENETRATING RAYS

By H. Geiger

Nature, London, vol. 127, No. 3212, 1931, pp. 785-787

In this article the author discusses the results obtained from experiments on the origin and nature of γ-radiation carried out by methods improved during the past few years in the following two directions: (1) along electrometric lines by the registration and the application of high-pressure chambers, and (2) by the method of electron counting. In the pressure chamber the degree of ionization produced in the gas by the ultra radiation is measured, while in the tube electron counter the high-velocity electrons coupled with the ultra radiation are registered singly.--W. Ayvazoglou.

(330) EINE EINFACHE METHODE ZUR AUTOMATISCHEN REGISTRIERUNG VON KOINZIDENZEN IN GEIGER-MÜLLERSCHEN ZÄHLROREN

(A SIMPLE METHOD FOR AUTOMATIC RECORDING OF COINCIDENCES IN GEIGER-MÜLLER'S TUBE COUNTERS)

By J. N. Hummel

Die Naturwissenschaften, Berlin, vol. 19, No. 18, 1931, pp. 375-376.

An arrangement by which two tube counters are electrically connected one with another in order to obtain direct coincidence measurements is described.

Owing to this arrangement the amplitude of the deflection of the electrometer is considerably increased.

The manner of connection is shown in a figure.--W. Ayvazoglou.

(331) STRAHLUNG VON ANTENNEN UNTER DEM EINFLUSS DER ERDBODENEIGENSCHAFTEN

(ANTENNAE RADIATION UNDER THE INFLUENCE OF SOIL PROPERTIES)

By M. J. O. Strutt

Annalen der Physik, Leipzig, vol. 9, No. 1, 1931, pp. 67-91.

In the first two sections of the article the author describes the arrangement and verification of a tubular sender and a tubular receiver provided with amplifier and a tubular voltmeter for 1.42 meters wave length. The main importance was laid to the work of the measurement arrangement exclusively by antennae, avoiding any parasitic radiation.

With this arrangement the electrical soil properties were measured by using 1.4 meters wave length according to a new method. A larger value of the index of refraction of the ground was obtained than that which results from the properties of the same soil measured at about 15 meters. The most probable explanation was thought to be the fact that the conductivity of the earth for very short waves (about 1 meter) increases considerably with the decrease of the length of the wave.

This behavior was established for the soil consisting of sand, humus, clay, and grassland. In case of water the measured index of refraction agreed with theoretical expectation (80 = $\mathcal{E}$).

Measurements concerning the direction of polarization of waves emitted from the vertical and horizontal antennae had also shown a great index of refraction of the earth. The theory could be verified by measuring radiation diagrams in a vertical plane; especially the theoretical expectation that the received field strength in vertical sending antennae increases close to the ground in proportion to the height above the ground. These measurements also had shown a large index of refraction of the earth. Finally, the dependency of the strength of receiving from the distance between the sender and receiver was measured for the vertical and horizontal sending antennae. Here the theoretical expectation, established first by von Hörschelman, according to which the distance law in case of the two kinds of antennae is the same, was confirmed.--Author's abstract translated by W. Ayvazoglou.

(332)2GEOLOGISCHE UND GEOPHYSIKALISCHE (RADIOACTIVE) UNTERSUCHUNGEN AM WESTRANDE DES GÖTTINGER LEINENTALGRABENS

(GEOLOGICAL AND GEOPHYSICAL (RADIOACTIVE) INVESTIGATIONS ON THE WESTERN BORDER OF THE GÖTTINGEN GRABEN OF THE VALLEY OF THE LEINE)

By Valeriu Patriciu

Abhandlungen der Preussischen Geologischen Landesanstalt, Berlin, No. 116, 1930, pp. 163-194.

This article consists of two parts; the first deals with the geological and the second with the geophysical (radioactive) investigation of the region mentioned in the title. Only the second part is considered in this abstract.

Contents of the second part:

1. Introduction. A brief historical outline of previous works on radioactive methods of investigation is given.

2. Apparatus. The apparatus used was that constructed by R. Ambronn (described in the Physikalische Zeitschrift, 1925, pp. 444-446); the method described in Ambronn's book, Methoden der Angewandten Geophysik, has been followed in general.

3. The plan of the work and the measurement procedure. The most practical way of carrying out the work is described; points of measurement to be selected along a straight line were fixed at about 8 meters distance one from another, the distances to be shortened to 4 or 2 meters only after abnormal distribution of emanation is observed. Holes of 3 centimeters in diameter and 80 to 100 centimeters in depth for inserting probe tubes are bored. The radium emanation of the ground is pumped into the ionization chamber. The reading of the position of the electrometer string is made every 30 seconds.

4. Preliminary tests to determine errors in the values of the measurements. The sources of these errors were eliminated by technical means.

5. Measurements for revealing the structure of the earth's crust. Plan of investigation; carrying out of measurements and graphical representation of the results; the results.

6. Limits of inaccuracy. The causes as well as the order of the errors were examined, and their influence upon the final results was established.

7. Concerning the reasons of higher radioactivity in tectonic fissures. Explanations are given.

In conclusion the author summarizes the results of geophysical investigation as follows:

1. The possibility of the application of the radioactive methods as developed by Ambronn for the explanation of tectonic questions in the regions of Saxony's disturbances is proved.

2. Material for the examination of the factors on which the success of the radioactive measurements depends is collected.

Maps of the region under investigation and a table with curves of radioactivities along the profiles are added.--W. Ayvazoglou.

(333) APPARATUS WITH CLOSED AIR CURRENT FOR THE DETERMINATION OF RADIOACTIVITY OF LARGE MINERALOGIC COLLECTIONS (IN RUSSIAN)

By V. Dubov and K. Pogodaev

Bulletin of the Geological and Prospecting Service, Leningrad, U.S.S.R., vol. 49, No. 9, 1930, pp. 100-101.

The apparatus described in this article is an improvement of a similar apparatus constructed by S. Artsybyshew and V. Pleshanova (see Geophys. Abs. No. 8, 1929, p. 18). The size of the metallic chamber in which the box containing the samples to be investigated is placed is reduced to 55 by 55 by 14 centimeters. The central part of the chamber has a raising of the form of a truncated cone, inside of which a ring-shaped commutator is fixed. Inside of the chamber, at 1.5 centimeters from its top and 4 centimeters from its walls,

a cover is placed; this cover is provided in the center with a tube 11 centimeters in diameter and 4 centimeters in height in which a propeller is inserted; the propeller is put in rotation by means of an electromotor.

The air driven by the propeller from the ionization chamber passes by the commutator, enters in the upper part of the apparatus, and returns again into the ionization chamber.

The current is measured by an electroscope provided with a reading microscope. Commutators of 9 to 13 centimeters in diameter and a capacity of 8.6 to 10 centimeters were used. Four disks coated with a thin layer of U_3O_8 were distributed over the bottom of the box of the size, 52 by 52 by 6 centimeters.

The results of observations are given in a table. A schematical design of the apparatus is shown in a figure.--W. Ayvazoglou.

6. GEOTHERMAL METHODS

(334) ZUSAMMENHANG ZWISCHEN DER GEOTHERMISCHEN TIEFENSTUFE UND DER WÄRMELEITFÄHIGKEIT DER GESTEINE

(RELATIONSHIP BETWEEN THE GEOTHERMAL DEPTH GRADIENT AND THE HEAT CONDUCTIVITY OF ROCKS)

By H. Börger

Glückauf, Essen, vol. 67, No. 17, 1931, pp. 545-551.

The article is divided into two chapters: (1) Investigation of the influence of the heat conductivity of rocks upon the geothermal gradient, and (2) heat-conductivity of various rocks.

From the investigations on the relationship between the heat conductivity of rocks and the geothermal gradient the author draws the conclusion that geothermal gradients have different values in different rocks deposited in layers and that their ratio depends on the heat conductivity of the different kinds of rocks.

The numerical value of the geothermal gradient inside of a certain type of rock depends not only on its capability of conducting heat but also on the ratio of magnitude of rocks overlying one another.

The author gives examples showing that in rocks, the geothermal gradient of which is high owing to their good heat conductivity, the temperature may be unusually high if the deposits overlying them have a lower heat conductivity. The differences in heat conductivity of single types of rocks are sufficient for the explanation of the unusually high temperatures often observed in salt mines and coal mines, without the necessity of assuming the existence of special sources of heat.

The article is illustrated by four figures.--W. Ayvazoglou.

(335) DEEP GOLD MINES AND GEOTHERMAL GRADIENTS

Editorial note

Engineering and Mining Journal, New York, vol. 131, No. 10, 1931, p. 447.

Geothermal gradients as established in some deep gold mines are mentioned in this note. The data may be of interest; they are:

1. The temperature at the 4,300-foot depth of the Kirkland Lake gold mine in Ontario is only 69° the increment being about 1° for every 150 feet.

2. In the St. John del Rey property, in Brazil, although the mine is at an altitude of 3,000 feet, the rock temperature at current working depths of below 7,100 feet is 127°, the geothermal gradient being about 1° for every 140 feet.

3. At the Kennedy, in California, the geothermal gradient was found to be 153 feet per degree, giving a rock temperature of about 91° on the 4,950-foot level.--W. Ayvazoglou.

7. UNCLASSIFIED METHODS

(336) APPLIED GEOPHYSICS

By A. Broughton Edge

Nature, London, vol. 127, No. 3212, 1931, pp. 783-785.

In this article the author gives a review of the recently published book entitled Applied Geophysics: A Brief Survey of the Development of Apparatus and Methods Employed in the Investigation of Subterranean Structural Conditions and the Location of Mineral Deposits, by Capt. H. Shaw, with the assistance of J. McG. Buckshaw and S. T. Newing (see Geophys. Abs. No. 25).

In the introduction to the review Edge says:

> In opening to the public a geophysical exhibition and in publishing a handbook which not only is descriptive of the exhibits but also constitutes an admirable historical résumé of and introduction to the study of applied geophysics, the Science Museum, South Kensington, has taken an important step in promoting the development of this branch of science, a step which is particularly appropriate at the present moment in view of the growing appreciation in Great Britain of the practical value of these relatively new methods of exploration.--W. Ayvazoglou.

(337) ARTICLES APPEARING IN THE TRANSACTIONS OF THE AMERICAN GEOPHYSICAL UNION, TWELFTH ANNUAL MEETING, APRIL 30, AND MAY, 1, 1931, HELD IN WASHINGTON, D. C.

National Research Council of the National Academy of Sciences, Washington, D.C., June, 1931.

The present volume includes the transactions of the twelfth annual assembly. The year's activities of the Union and on the Stockholm meeting and on proposed geophysical work in polar regions and resolutions submitted by the several sections were adopted. The individual section meetings were marked by numerous progress reports on the geophysical activities not only in the United States but also in Canada and Mexico.

The first part of the volume (pp. 7-34) deals with reports and papers presented to the general assembly. The contents are as follows:

1. Resolutions adopted: On gravity at sea; on internal cooperation in the study of tidal waves; on comparisons of new types of seismological instruments developed in the U. S. with various types developed in Europe; commemorating fiftieth anniversary of General Greely's participation in the First International Polar Year; and on the death of Franklin G. Tingley.

2. A short report on the Stockholm Assembly and on proposed geophysical work in polar regions, by J. A. Fleming.

3. Symposium on time signals: (a) United States Naval Observatory time-service, by J. F. Hellweg; (b) Time signals for electrical and physical measurements, by Frank Wenner; (c) Time signal needs for geodetic work, by Edwin J. Brown; (d) Letters bearing on symposium from J. B. Macelwane, Harry O. Wood, and B. Gutenberg; (e) Establishment of world time, by F. W. Lee; (f) The service available from the standard-frequency transmissions of the Bureau of Standards, by J. H. Dellinger; (g) The accuracy of the primary frequency standard of the Bureau of Standards, by Charles G. McIlwraith; and (h) Informal communications by C. W. Horn, H. A. Affel and W. A. Marrison.

In the following parts of the volume the papers presented in various sections are discussed:

The Section of Geodesy (pp. 35-58). Gravity work, instrumental progress, the moon's influence on latitude, and the establishment astronomically of points on an unsurveyed line were presented in the following papers: (a) Progress report on the absolute determination of gravity at Washington, D.C., by Paul R. Heyl; (b) Mexican gravity stations in 1930 and first calculations and corrections for topography and isostasy, by Elfego Ruiz; (c) Gravity at the Stockholm meeting of the International Geodetic and Geophysical Union, by Walter D. Lambert; (d) Progress report on graduation and calibration of precision-circles, by Lewis V. Judson; (e) Further investigations of the moon's influence on latitude, by H. T. Stetson; (f) The establishment astronomically of points on an unsurveyed provincial boundary, by Noel J. Ogilvie; and (g) Progress in geodetic work during the past year, by William Bowie.

The Section of Seismology (pp. 59-78). Attention was devoted to theoretical considerations, relations of geodetic operations to seismological interpretations, registration of time signals, progress reports on instrumental developments, and the provision made to realize the Union's resolution at its eleventh annual assembly to establish a seismological observatory in South America (near Huancayo, Peru). The following papers were presented in connection with these questions: (a) Velocity of explosion-generated longitudinal waves in a nepheline syenite, by L. Don Leet and Maurice Ewing; (b) Geodetic work lays the basis for study of earth movements, by William Bowie; (c) Registration of the time signals at Georgetown, by F. W. Sohon; (d) The origin of earthquake waves, by Harry Reid; (e) Report on proposed establishment of a seismological observatory at Huancayo, Peru (read by the secretary); (f) Report of progress on accelerometer for recording earthquake shocks, by G. L. Hosmer; (g) Progress report on development of instruments, by Frank Wenner and McComb; (h) The development of a tilt-meter, by George E. Merritt; (i) Progress report on development of seismological instruments, by H. E. Comb; (k) Report on the construction of a 3-drum seismograph recorder, by W. H. Reynolds; (l) Observations of development in instruments at the Seismological Research Laboratory in Pasadena, by N. H. Heck; and (m) Memorandum on the New York earthquake of April 20, 1931, by Frank Neumann.

The Section of Meteorology (pp. 79-90). Much time was given to discussion of the international polar year prospects, followed by a report on the Madrid meeting in March, 1931, of the international commission for the exploration of the upper air, discussion of proposed international cloud atlas and papers on atmospheric turbidity, measurement of color of sea and sky, and cyclical variations in relation to long-range weather forecasting. Nine papers were presented.

Section of Terrestrial Magnetism and Electricity (pp. 91-144). The papers presented to this section were given in four groups relating to (1) the year's investigations in progress in the United States; (2) the Stockholm assembly, (3) extraterrestrial considerations, and (4) polar research. The list of the papers is as follows: (a) magnetic work of the United States Coast and Geodetic Survey, 1930-31, by D. L. Hazard; (b) Field and laboratory investigations of the Carnegie Institution of Washington, by J. A. Fleming; (c) Recent developments in radio-transmission measurements, by G. W. Kenrick and G. W. Pickard; (d) The U. R. S. I. cosmic radio broadcast, by Watson Davis; (e) Reports of geomagnetic and electric work of organizations and individual investigators in the United States, 1930-31, by Harlan W. Fisk; (f) Reports submitted on work in terrestrial magnetism and atmospheric electricity (fourteen papers); (g) The Stockholm assembly of the International Union of Geodesy and Geophysics--Proceedings of the section of Terrestrial Magnetism and Electricity, by H. D. Harradon; (h) Investigations at the Perkins Observatory of changes in the Kennelly-Heaviside layer as a function of lunar altitudes, by H. T. Stetson; (i) Notes on correlation investigations between Kennelly-Heaviside layer and lunar altitudes, by Greenleaf W. Pickard; (k) Use of magnetic data for investigating radiation from the sun, by J. Bartels; (l) The electrical state of the sun, by Ross Gunn; (m) The ultraviolet-light theory of comet activity, by H. B. Maris; (n) Magnetic observations on a moving ice floe, by W. J. Peters; (o) Isomagnetic charts of the arctic area, by Harlan W. Fisk; (p) Significance of geoelectric data from the polar regions, by O. H. Gish; and (q) A Fort Conger polar expedition program, by H. B. Maris.

The Section of Oceanography (pp. 145-180). The communications presented to this section were chiefly progress reports showing excellent development of oceanographical work in the United States by governmental and private organizations (11 papers).

Section of Volcanology (pp. 181-186). Five papers were given relating to the solubility of water in granite magmas, and to the Merapi, Katmai district, Cripple Creek, and Valles Mountain volcanoes.

Section of Hydrology (pp. 187-229). The meeting of this section was devoted to statements of the formal organization of the section (14 papers were presented).--W. Ayvazoglou.

(338) ENTWICKLUNG UND STAND DER GEOPHYSIKALISCHEN DURCHFORSCHUNG DER SÜDSTAATEN VON U. S. A.

(DEVELOPMENT AND CONDITION OF GEOPHYSICAL EXPLORATION OF THE SOUTHERN STATES OF THE U. S. A.)

By Rudolf v. Zwerger

Petroleum Zeitschrift, Berlin, vol. 27, No. 19, 1931, pp. 335-347.

The author divides the southern States, the development and conditions of geophysical exploration of which is discussed in this article, into four large regions:

1. The Gulf embayment.
2. Bend Arch and Balcones fault zone.
3. Permian Basin of west Texas and New Mexico.
4. The Amarillo-Wichita-Arbuckle mountains with the bordering parts of north Texas and south Oklahoma.

The use of various geophysical methods with respect to these regions is discussed. The difference in geological conditions and their effect upon the development of geophysical work during the last years is mentioned.

In conclusion the author gives a prospect on the future possibilities of geophysical exploration in the southern States.--W. Ayvazoglou.

(339) PROGRÈS DE LA GÉOPHYSIQUE APPLIQUÉE

(PROGRESS OF APPLIED GEOPHYSICS)

By J. B. Ostermeier-Althegnenberg

La Revue Pétrolifère, Paris, Nos. 425, 426, 427, 1931, pp. 655-659, 685-688, and 717-720.

The original article was published by the author in the Internationale Zeitschrift für Tiefbohrtechnik, Erdölbergbau und Geologie, vol. 38, 1930. The article appearing in the Revue Pétrolifère is a translation made by Pierre Pelecier.

Magnetic, gravimetric, seismic, and electrical methods of prospecting are discussed and illustrated by examples; figures and diagrams are added. In the last part of the article the author gives a description of improvements made by him on the field balance (Askania) in order to increase the precision and rapidity in measurements of the vertical terrestrial magnetic intensity. The problem of lessening the influence of the ground by elevating the instrument is partly solved by providing the instrument with a horizontal instead of a vertical visual apparatus; some other improvements are described, especially those concerning the influence of variations of temperature. The time necessary for one observation was reduced to about 7 minutes, as compared with 15 to 20 minutes required by the instruments used so far.--W. Ayvazoglou.

(340) GEOPHYSICAL INVESTIGATIONS CARRIED OUT IN THE SALT-DOME AREAS OF THE GULF COAST OF TEXAS AND LOUISIANA

By Carl Schmidt.

The Petroleum Times, London, vol. 25, No. 638, 1931, pp. 508-509.

Some of the results which had been derived from the use of seismic methods, either alone or in conjunction with the gravimetric method, in the salt-dome areas of the Gulf Coast of Texas and Louisiana since 1924 are described and the most important discoveries mentioned. Schmidt concludes that the extensive use of seismic method, as well as other geophysical methods of investigation, had resulted in the finding and exploitation of oil fields of enormous value, enabled oil companies to avoid the expenditure of vast sums of money in fruitless drilling, and had made it possible for operators to plan successful and economical drilling campaigns for many years ahead.--W. Ayvazoglou.

(341) THE PROGRESS OF GEOPHYSICS

Editorial note

The Petroleum Times, London, vol. 25, No. 638, 1931, p. 510.

This article refers to the exhibition at the Science Museum in London of apparatus and equipment used in geophysics.

Many types of instruments now in use for geophysical prospecting by magnetic, gravitational, and seismic methods are represented. A special brochure to assist students to get maximum value from a study of the exhibits has also been prepared.--W. Ayvazoglou.

(342) SEARCHING FOR MINERALS WITH SCIENTIFIC INSTRUMENTS

By E. Lancaster-Jones

Journal of Scientific Instruments, London, vol. 8, No. 2, 1931, pp. 34-44.

A series of physical instruments recognized as valuable aids in the search both for mineral deposits directly, and for structural features which may be favorable to mineral accumulation is described.

The instruments are divided into four main groups, characterized as gravitational, magnetic, seismic, and electrical, in relation to the particular physical magnitudes of which each group serves to measure the variation.

1. Gravity method. The Eötvös torsion balance, the balance improved by L. Örtling, and that improved by the author and Captain Shaw (manufactured by the Cambridge Instrument Co., Ltd.) are described and photographs of them are given.

2. Magnetic method. Dip needle devised by Lloyd, Schmidt's portable vertical variometer, and a portable horizontal variometer devised by Walker are mentioned.

3. Seismic method. Mechanisms for amplifying a vibration are explained and the question as to how this type of apparatus can be used in prospecting is considered. A seismograph manufactured by the Cambridge Instrument Co. is demonstrated.

4. Electrical method. Three methods (potential, resistivity, and electromagnetic) are discussed, and the apparatus used is explained.--W. Ayvazoglou.

8. GEOLOGY

(343) THE SALT-DOME AREA WEST OF CELLE, GERMANY

By Carl Schmidt

The Petroleum Times, London, vol. 25, No. 638, 1931, p. 508.

Notwithstanding the numerous boreholes that had been sunk, very little was known about the actual size and shape of the two salt domes (Wietze and Hambühren) of the north German plain. The questions whether these domes were separated from each other or whether there was a connection between them had not been determined. In order to decide this question seismographical investigations were carried out. The question for determining the actual size and shape of the domes had been solved by seismic, as well as gravimetric, investigations.

These investigations carried out in the area of Wietze-Hambühren-Wittbeck-Wolthausen-Meissendorf revealed unexpected and very important geological conditions. It was determined that:

1. Instead of there being two domes there was in fact only one large dome.

2. That part of the dome around Hambühren was much larger and had an entirely different shape than was anticipated.

3. In the direction of the south southwest-north northeast prolongation of the Hambühren dome there was another salt upthrust.

4. The dome of Meissendorf at its southern part had an elongation toward the southeast in the east southeast-west northwest direction.

The results of geophysical investigations also confirmed the importance of the two tectonic lines of dislocation, in directions mentioned in 3 and 4, in the underground structures of the north German plain, and showed that the occurrence of salt upthrusts was connected with such lines of tectonic dislocation, especially with the intersections of the predominant striking directions.--W. Ayvazoglou.

9. NEW BOOKS

(344) Edge, A. B. Broughton, and Laby, T. H. Principles and practice of geophysical prospecting. Edited by Edge and Laby. Pp. XIV + 380. With 261 illustrations in the text. Price, 15 s. net. Cambridge University Press, London, Fetter Lane, E. C. 4; New York, Toronto, Bombay, Calcutta, Madras. MacMillan, 1931. A report of the Imperial Geophysical Experimental survey.

(345) Emmons, William Harvey. Geology of petroleum. 736 pages, 435 illustrations. McGraw-Hill Book Co., 1931, New York City. Price, $6. The work may be broadly divided into two sections. One of these deals with the general principles of oil geology. The second section describes the features of the main oil fields of the world, and gives innumerable references. The book is profusely illustrated.

(346) Kaiser, August. Beitrage zur Physikalischen Erforschung der Erdrinde, Heft 2: Magnetische Messungen in Nordwestdeutschland. (Report on physical research in the earth crust. Part 2: Magnetic measurements in northwest Germany.) 50 pages, 3 tables, 5 figures. Issued by the Preussische Geologische Landesanstalt, Berlin, 1930, Price, R.M. 7.50.

10. PATENTS

(347) GRAVITATIONAL BALANCE

K. D. Sabaneev

Russian patent 14,274 — Patent issued March 31, 1930.

The gravitational balance described in this patent is put under a bell from which the air can be partly exhausted; a special arrangement for reading is made and a polished ball, serving as an artificial illuminating point, is used for photographic recording of the indications of the balance.

Claims allowed - 3.

(348) SEISMOGRAPH

G. M. Polonsky and V. G. Leshchinsky

Russian patent 15,527 — Patent issued May 31, 1930.

The construction of the seismograph disclosed in this patent is characterized by the application of a Rochelle salt plate secured in an isolated holder and connected by one end with the net of an amplifying lamp and by the other end with the ground and one pole of the battery producing the glowing of the lamp, in the anode circuit of which an electric recording apparatus is inserted.

Claims allowed - 1.

(349) APPARATUS FOR TESTING THE RADIOACTIVITY OF ROCKS

M. J. Sumgin

Russian patent 16,787 Patent issued Sept. 30, 1930.

The apparatus described in this patent and assigned for testing the radioactivity of rocks consists of a hollow cylinder in the lower end of which a movable bottom of the form of a truncated cone, turned with its top downward, is inserted; the walls of this cone are covered with a thin layer of zinc sulphide or platinocyanide barium; the bottom has an opening in which a plate assigned for carrying the sample of the rock to be investigated is inserted; the upper end of the cylinder has a cover provided with a microscope serving for observation.

Claims allowed - 1.

(350) METHOD OF CONTOURING ORE DEPOSITS

L. N. Bogoiavlensky

Russian patent 12,953 Patent issued Jan. 31, 1930.

The method of contouring ore deposits disclosed in this patent is characterized by the following proceeding: At the points of a net plotted on the terrane, the "half value period" of the decay of any radioactive matter is determined, then the points with equal values of these points are connected by curves; based on the form of the curves having equal "half value periods" of the decay, the contour of the ore deposit by which the change of the half value period of the decay is caused is drawn on a plane.

Claims allowed - 1.

(351) METHOD OF PROSPECTING FOR ORE

A. G. Tarkhov

Russian patent 17,515 Patent issued Sept. 30, 1930.

The method described in this patent in which heterodine is used, based on its property of changing one of its contours under the influence of ore deposits, is characterized by an arrangement such that the antenna connected with one of the contours of the heterodine is placed inside of a metallic screen, the latter being provided with a slot. In adjusting the apparatus the slot is covered with a metallic plate. The screen can be rotated by means of a device operated by hand. After the adjustment of the apparatus for the investigation of the ground (disappearing of sound in the telephone) the slot is opened and the screen rotated. The appearance of a sound in the telephone shows that in the direction of the slot there is an ore deposit.

Claims allowed - 2.

INDEX[3]

3 The first figure refers to the number of the abstract, the second to the method of prospecting as indicated in the table of contents, and the third to the page.

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

LIST OF MOTORS AVAILABLE TO PROSPECTIVE BUILDERS OF PERMISSIBLE OUTFITS

BY

L. C. ILSLEY AND M. W. MEANS

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

LIST OF MOTORS AVAILABLE TO PROSPECTIVE BUILDERS OF PERMISSIBLE OUTFITS[1]

By L. C. Ilsley[2] and M. W. Means[3]

The United States Bureau of Mines under provisions of Schedule 2C[4] issues approvals covering complete operating units such as loading machines, room hoists, and storage-battery locomotives. Certain manufacturers design and build entire machines. Other manufacturers purchase a part or all of the electrical accessories from manufacturers of electrical equipment and assemble them, thereby making up a permissible unit.

Before the assembler purchases electrical accessories for use in such contemplated units he may require certain information from the accessory manufacturer as to whether these particular accessories are suitable for assembly in his proposed permissible outfit. In order to aid accessory manufacturers in marketing their product, a provision was incorporated in Schedule 2C whereby tests might be made on individual accessories and reports rendered as to the suitability of a given accessory for use in permissible assemblies.

The section of Schedule 2C which permits this special work reads as follows:

CONDITIONS UNDER WHICH SEPARATE PARTS WILL BE INVESTIGATED AS TO SUITABILITY FOR INCORPORATION ON PERMISSIBLE MACHINES

Manufacturers having a motor, controller or other separate electrical units which they wish to offer to a prospective builder of a permissible machine may submit such units for inspection and test and obtain a report on the results from the bureau. (See Inspection and Test Reports, page 23).

Application

A letter accompanied by appropriate fees and requesting an investigation into the qualifications of the unit for incorporation in permissible machines, should be addressed to the Director, Bureau of Mines, Washington, D. C.

Inspections and Tests

The procedure followed by the bureau in making inspections and tests of units to determine their suitability is identical with that followed for parts of permissible equipment as previously outlined in this schedule. The fees, drawings, and other details to be taken care of by the manufacture are likewise the same for separate units as for parts of complete machines submitted for approval.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6548."

2 - Supervising engineer, U. S. Bureau of Mines, electrical section, Pittsburgh Experiment Station, Pittsburgh, Pa.

3 - Assistant electrical engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

4 - U. S. Bureau of Mines, Explosion-Proof Mine Equipment: Schedule 2C, February, 1930, 27 pp.

Report on Suitability

When a separate unit has passed the required inspections and tests and suitable drawings have been filed with the bureau a formal letter of notification that the unit has the necessary qualifications will be furnished the manufacturer from the Washington office of the bureau. This letter will authorize the manufacturer to offer such separate unit to a builder of permissible machines with the assurance that, unless modified further, inspection and test of the unit will not be required by the bureau. The use of the bureau's name on any label or plate attached to the separate unit, or advertising the unit as "approved," will not be sanctioned.

Note: It should be understood that a Bureau of Mines approval covers complete machines only, and therefore approval of machines in which separate units are installed can not properly be claimed by the assembler or builder without first obtaining formal authorization from the bureau.

Motors that have been satisfactorily reported upon in accordance with the foregoing procedure, or motors that have already been used in permissible assemblies can be offered to prospective customers with the assurance that if furnished in the exact design previously tested, these motors will not need to be retested in order to obtain approval of a permissible assembly of which the motor constitutes a part.

It often happens, however, that a customer wants a motor slightly different from the one previously tested and inspected by the bureau. If the change is such as to alter the unoccupied space (internal volume) of the motor, affect the flange joints of the motor frame or covers, or modify the flame path through the bearings or along the lead entrance, a retest of the motor will in most cases be deemed necessary. Also a change in material or a modification of the bolt spacing may require a retest. Slight changes in windings or in the length of rotors in a given frame construction can sometimes be approved from an examination of the drawings showing the difference between the old and new design.

The following lists of D.C. and A.C. motors have been prepared in order that prospective purchasers may have a ready list of motors that have been judged suitable for incorporation in permissible outfits. These lists have been made up on a basis of continuous rating, as including the intermittent ratings rendered the lists too complicated. However, in most cases the motors can be furnished with windings suitable for intermittent duty.

Direct-Current Motors[1]

Hp.	R.p.m.	Voltage[2]	Type or frame No.	Manufacturer
1/2	850	230	M-1	Star Electric Co.
3/4	850	do.	DF-26	Continental Elec. Co.
3/4	850	do.	EF	Crocker-Wheeler.
3/4	850	do.	M-2	Star Electric Co.
3/4	850	do.	RH	Westinghouse E. & M. Co.
3/4	1150	do.	M-1	Star Electric Co.
1	850	do.	D-325	Continental Elec. Co.
1	1150	do.	DF-26	Continental Elec. Co.
1	1150	do.	EF	Crocker-Wheeler.
1	1150	do.	BD-226	General Electric Co.

1 - A wide range of intermittent ratings are available on the frame sizes listed.

2 - Most of the above motors can be furnished for use on 500-volt circuits.

Direct-Current Motors[1] - Continued

Hp.	R.p.m.	Voltage[2]	Type or frame No.	Manufacturer
1	1150	230	M-2	Star Electric Co.
1	1750	do.	DF-23	Continental Elec. Co.
1	1750	do.	BD-226	General Electric Co.
1	1750	do.	RZ	Reliance Elec. & Eng. Co.
1	1750	do.	M-1	Star Electric Co.
1-1/4	1150	do.	M-2-1/2	Star Electric Co.
1-1/4	1150	do.	RH	Westinghouse E. & M. Co.
1-1/2	850	do.	D-327	Continental Elec. Co.
1-1/2	850	do.	GFB	Crocker-Wheeler.
1-1/2	1150	do.	D-325	Continental Elec. Co.
1-1/2	1150	do.	BD-232	General Electric Co.
1-1/2	1150	do.	XM-2	Star Electric Co.
1-1/2	1750	do.	DF-26	Continental Elec. Co.
1-1/2	1750	do.	EF	Crocker-Wheeler.
1-1/2	1750	do.	BD-226	General Electric Co.
1-1/2	1750	do.	M-2	Star Electric Co.
1-3/4	1150	do.	XM-2-1/2	Star Electric Co.
1-3/4	1750	do.	M-2-1/2	Star Electric Co.
2	850	do.	DF-45	Continental Elec. Co.
2	850	do.	33-SK	Westinghouse E. & M. Co.
2	1150	do.	D-327	Continental Elec. Co.
2	1150	do.	GFB	Crocker-Wheeler
2	1150	do.	BD-234	General Electric Co.
2	1750	do.	D-325	Continental Elec. Co.
2	1750	do.	BD-232	General Electric Co.
2	1750	do.	XM-2	Star Electric Co.
2	1750	do.	RH	Westinghouse E. & M. Co.
2-1/2	1750	do.	XM-2-1/2	Star Electric Co.
3	850	do.	DF-47	Continental Elec. Co.
3	850	do.	53-SK	Westinghouse E. & M. Co.
3	1150	do.	DF-45	Continental Elec. Co.
3	1150	do.	CD-65	General Electric Co.
3	1150	do.	33-SK	Westinghouse E. & M. Co.
3	1750	do.	D-327	Continental Elec. Co.
3	1750	do.	GFB	Crocker-Wheeler.
3	1750	do.	CD-65	General Electric Co.
3	1750	do.	M-4	Star Electric Co.
3	1750	do.	33-SK	Westinghouse E. & M. Co.
5	850	do.	DF-49	Continental Elec. Co.
5	850	do.	63-SK	Westinghouse E. & M. Co.
5	1150	do.	DF-47	Continental Elec. Co.
5	1150	do.	CD-73	General Electric Co.

1 - A wide range of intermittent ratings are available on the frame sizes listed.

2 - Most of the above motors can be furnished for use on 500-volt circuits.

Direct-Current Motors[1] - Continued

Hp.	R.p.m.	Voltage[2]	Type or frame No.	Manufacturer
5	1150	230	53-SK	Westinghouse E. & M. Co.
5	1750	do.	DF-47	Continental Elec. Co.
5	1750	do.	CD-73	General Electric Co.
5	1750	do.	53-SK	Westinghouse E. & M. Co.
7-1/2	850	do.	CD-95	General Electric Co.
7-1/2	1150	do.	DF-49	Continental Elec. Co.
7-1/2	1150	do.	63-SK	Westinghouse E. & M. Co.
7-1/2	1750	do.	DF-49	Continental Elec. Co.
10	850	do.	CD-95	General Electric Co.
10	850	do.	103-SK	Westinghouse E. & M. Co.
10	1150	do.	CCM	Reliance Elec. & Eng. Co.
10	1150	do.	103-SK	Westinghouse E. & M. Co.
10	1750	do.	63-SK	Westinghouse E. & M. Co.
15	850	do.	113-SK	Westinghouse E. & M. Co.
15	1150	do.	CD-95	General Electric Co.
15	1150	do.	113-SK	Westinghouse E. & M. Co.
15	1750	do.	CD-95	General Electric Co.
15	1750	do.	103-SK	Westinghouse E. & M. Co.
20	850	do.	111-SK	Westinghouse E. & M. Co.
25	1150	do.	111-SK	Westinghouse E. & M. Co.

1 - A wide range of intermittent ratings are available on the frame sizes listed.

2 - Most of the above motors can be furnished for use on 500-volt circuits.

Alternating-Current Motors

Hp.	R.p.m.	Voltage	Type or frame No.	Manufacturer
1/2	600	220-440	016	Howell Electric Motors Co.
1/2	720	do.	008	Howell Electric Motors Co.
1/2	900	do.	004	Howell Electric Motors Co.
1/2	1200	do.	004	Howell Electric Motors Co.
1/2	1200	do.	24T	The Louis Allis Co.
1/2	1800	do.	004	Howell Electric Motors Co.
3/4	514	do.	43N	The Louis Allis Co.
3/4	600	do.	105	Howell Electric Motors Co.
3/4	720	do.	016	Howell Electric Motors Co.
3/4	900	do.	008	Howell Electric Motors Co.
3/4	900	do.	205Z	The Louis Allis Co.
3/4	1200	do.	008	Howell Electric Motors Co.
3/4	1800	do.	004	Howell Electric Motors Co.
1	600	do.	111	Howell Electric Motors Co.

Alternating-Current Motors - Continued

Hp.	R.p.m.	Voltage	Type or frame No.	Manufacturer
1	600	220-440	43N	The Louis Allis Co.
1	720	do.	105	Howell Electric Motors Co.
1	900	do.	016	Howell Electric Motors Co.
1	1200	do.	016	Howell Electric Motors Co.
1	1200	do.	24T	The Louis Allis Co.
1	1800	do.	008	Howell Electric Motors Co.
1-1/2	600	do.	124	Howell Electric Motors Co.
1-1/2	720	do.	111	Howell Electric Motors Co.
1-1/2	720	do.	43N	The Louis Allis Co.
1-1/2	900	do.	105	Howell Electric Motors Co.
1-1/2	1200	do.	024	Howell Electric Motors Co.
1-1/2	1200	do.	205Z	The Louis Allis Co.
1-1/2	1800	do.	016	Howell Electric Motors Co.
1-1/2	1800	do.	24T	The Louis Allis Co.
2	450	do.	66P	The Louis Allis Co.
2	600	do.	132	Howell Electric Motors Co.
2	720	do.	124	Howell Electric Motors Co.
2	900	do.	111	Howell Electric Motors Co.
2	900	do.	43N	The Louis Allis Co.
2	1200	do.	105	Howell Electric Motors Co.
2	1800	do.	024	Howell Electric Motors Co.
2-1/2	514	do.	66P	The Louis Allis Co.
3	720	do.	132	Howell Electric Motors Co.
3	900	do.	124	Howell Electric Motors Co.
3	1200	do.	111	Howell Electric Motors Co.
3	1200	do.	43N	The Louis Allis Co.
3	1800	do.	105	Howell Electric Motors Co.
5	720	do.	66P	The Louis Allis Co.
5	900	do.	132	Howell Electric Motors Co.
5	1200	do.	124	Howell Electric Motors Co.
5	1800	do.	111	Howell Electric Motors Co.
5	1800	do.	43N	The Louis Allis Co.
7-1/2	600	do.	K404	General Electric Co.
7-1/2	900	do.	66P	The Louis Allis Co.
7-1/2	1200	do.	132	Howell Electric Motors Co.
7-1/2	1800	do.	124	Howell Electric Motors Co.
10	600	do.	K405	General Electric Co.
10	720	do.	K404	General Electric Co.
10	1200	do.	66P	The Louis Allis Co.
10	1800	do.	132	Howell Electric Motors Co.
15	720	do.	K405	General Electric Co.
15	900	do.	K404	General Electric Co.
20	900	do.	K405	General Electric Co.
20	1200	do.	K404	General Electric Co.
25	1200	do.	K405	General Electric Co.
30	1800	do.	K405	General Electric Co.

I. C. 6549 January, 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

PHYSICAL CHEMICAL PROPERTIES OF METHANE

BY

H. H. STORCH

I.C.6549.
January, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

PHYSICAL CHEMICAL PROPERTIES OF METHANE[1]

By H. H. Storch[2]

INTRODUCTION

Methane is the chief constituent of natural gas, and large quantities of it are also obtained in coke-oven gas and in the off-gases from cracking higher hydrocarbons. Perhaps the most important potential chemical use of methane is in the production of higher hydrocarbons, both aliphatic and aromatic. The results of a relatively large number of researches have been published in the technical and patent literature on the thermal decomposition of methane, the objective being an industrial process for the production of "antiknock" motor fuel. No industrially feasible process has, however, been developed. An inspection of the literature on this subject indicates that the chief difficulty in the way of accomplishing this object lies in controlling the decomposition so that undesirable products are eliminated. A considerable number of quantitative experiments have been reported, the results of which provide only a vague and confusing picture of the various steps in the thermal decomposition of methane. The reaction mechanism has not been subjected to rigorous fundamental study, and the necessity for such a study is obvious if further progress is to be made.

The thermal decomposition is only one of the reactions of methane which are of potential industrial importance and which require further fundamental investigation. The oxidation, hydrolysis, and halogenation of methane are other reactions which must be studied further. In view of the importance of methane as a raw material for various industrially desirable products, it is appropriate that a survey be made of the important facts concerning the physical and chemical properties of methane which will serve as an aid in subsequent experimental work. No attempt will be made to present a complete chronological review, but rather a condensed critical description of the important literature will be presented.

THERMODYNAMIC PROPERTIES OF METHANE

Keyes, Smith, and Joubert[3] and Keyes and Burks[4] studied the pressure-volume relationship for CH_4 in the pressure range 1 to 300 atmospheres and 273 to 473° Abs. They findthe following equation of state to represent their data accurately:

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6549."

2 Principal physical chemist, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

3 Keyes, F. G., Smith, L. B., and Joubert, D. B., The Equation of State for Methane Gas Phase: Jour. Math. and Phys., Mass. Inst. Tech., vol. 1, 1922, p. 191.

4 Keyes, F. G., and Burks, H. G., The Isometrics of Gaseous Methane. Jour. Am. Chem. Soc., vol. 49, 1927, p. 1403.

$$p = \frac{5.1173\,(T - y/2)}{v - d} - \frac{9370\,(1 - 0.586\,y)}{v + 0.42}$$

$$\log_{10} d = 0.5611 - 0.978/v$$

$$\log_{10} (v - d) \frac{y}{1 - y^2} = \frac{527}{T} - 1.25 \log_{10} T + 0.813$$

where p = pressure in atmospheres and
T = temperature in degrees absolute.

Keyes, Taylor, and Smith[5] measured the vapor pressure of liquid CH_4 from 95 to 191° A., and they present the following equation therefor:

$$\log_{10} p = -\frac{595.546}{T} + 8.09938 - 4.04175 \times 10^{-2}\,T + 1.68655 \times 10^{-4}\,T^2 - 2.51715 \times 10^{-7}\,T^3$$

where p is the vapor pressure in atmospheres, and
T the temperature in degrees absolute.

The boiling point of CH_4 is given by these authors as 111.52° A. By means of the above vapor pressure equation and measurements of the densities of the coexisting phases, Keyes and his coworkers calculated latent heats of vaporization by means of the Clapeyron equation and represented the results by the following equation:

$$\log_{10} L = 1.65214 + 2.0076 \times 10^{-4}(T_C - T) + 0.2225 \log_{10} (T_C - T)$$

where L = heat of vaporization in calories per gram, and
T_C = critical temperature (191.030 A.).

The heat of vaporization at the boiling point (111.52° A.) is, according to this equation, 123 calories per gram or 1968 calories per mol. This figure is considerably at variance with that calculated by the writer from the vapor pressure measurements of Stock, Henning, and Kuss[6] -- namely, 2,075 calories per mol. This equation, however, yields 2,104 calories per mol. for the heat of vaporization at the triple point (89.98° A.), which is in excellent agreement with calculations based on the vapor pressure of solid methane as measured by Karwat[7] in the range 76.89 to 87.25° A. and by Freeth and Verschoyle[8] in the range 64.78 to 90.66° A., and on the heat of fusion as given by Clusius.[9] The heat of vaporization of solid methane as calculated from Karwat's measurements is 2,326 calories per

5 Keyes, F. G., Taylor, R. S., and Smith, L. B, The Thermodynamic Properties of Methane: Jour. Math. and Phys., Mass. Inst. Tech., vol. 1, 1922, p. 211.

6 Stock, A., Henning, F. and Kuss, E., Dampfdrucktafelen für Temperaturbestimmung zwischen 25° und -185°: Ber. Deut. Chem. Gesell., vol. 54, 1921, p. 1119.

7 Karwat, F., Die Dampfdruck des festen Chlorwasserstoffs, Methans, und Ammoniaks. Ztschr. Physik. Chem., vol. 112, 1924, p. 486.

8 Freeth, F. A., and Verschoyle, T. T. H., Physical Constants of the System Methane-Hydrogen: Proc. Roy. Soc. (London) vol. A 130, 1931, p. 453.

9 Clusius, K., Uber die spezifische Warme Einiger kondensierter Gase zwischen 10° abs. und ihrem Triple punkt: Ztschr. physik. Chem., vol. B 3, 1929, p. 41.

mol. which checks very closely with a similar calculation using Freeth and Verschoyle's data. Combining 2,326 calories per mol. with the heat of fusion (at 90.60 A.) of 224 calories per mol. as given by Clusius[9], the value 2,102 calories per mol. is obtained for the heat of vaporization of liquid methane in the vicinity of the melting point. This figure checks that of Keyes, Taylor, and Smith[5] within 0.1 per cent.

The specific heat of methane from 10.33 to 105.3° A. has been recently measured by Clusius. The specific heat equation given by Eastman[10] for CH_4 may be used in the temperature interval from the boiling point to 298° A. Three determinations of the specific heat in the range 297.7 to 481.2° A. are given by Eucken and Lüde[11], and they are probably more accurate than the data presented by Dixon, Campbell, and Parker[12] which are based on determinations of the velocity of sound in CH_4 at high temperatures. Ludolph[13] presents specific-heat data calculated from theoretical considerations of the structure of the methane molecule. His results are all appreciably lower than any of the experimental data.

A calculation[14] of the entropy of CH_4 based on the specific-heat data outlined in the preceding paragraph yielded the value 43.91 E.U. at 25° C. Calculations of the entropy of CH_4 by way of equilibrium measurements for the reactions:

$$C + 2H_2 \rightleftarrows CH_4 \qquad (1)$$
$$CH_4 + H_2O \rightleftarrows CO_2 + 4H_2 \qquad (2)$$

yielded the values 45.24 and 42.16 entropy units, respectively. Villars[15] computed three values for the entropy of CH_4 from spectroscopic data--namely, 44.1, 42.3 and 42.0, indicating a preference for the last figure. The value 43.91 entropy units calculated from low-temperature specific-heat data is considered by the writer to be the most reliable estimate available at present. A free energy equation based on this entropy figure, and on the heat of combustion of CH_4 recently measured by Rossini[16] using specific heat equations for C, H_2, and CH_4 as indicated elsewhere[17] by the writer, is as follows:

$$\Delta F = - 15{,}313 + 10.54\, T\ln T - 4.36 \times 10^{-3}T^2 - 0.11 \times 10^{-6}T^3 - 48.1T$$

CHEMICAL REACTIONS

Methane is a relatively inert gas so far as chemical activity is concerned. This inertness toward chemical change is to be correlated with evidence from ionization potential experiments which indicate that the outer "shell" of the methane molecule consists of 8 electrons similar to the arrangement in the rare gases.[18]. That is to say, the hydrogen atoms

10 Eastman, E. D., The Specific Heats of Gases at High Temperatures: Tech. Paper 445, Bureau of Mines, 1929, 27 pp.

11 Eucken, A., and Lüde, K. V., Die Spezifische Waerme der Gase bei mittleren und hohen Temperaturen: Ztschr. physik. Chem., vol. B 5, 1929, p. 436.

12 Dixon, H. B., Campbell, C., and Parker, A., On the Velocity of Sound in Gases at High Temperatures and the Ratio of the Specific Heats: Proc. Roy. Soc. (London), vol. A 100, 1921, p. 1.

13 Ludolph, P. C., The Specific Heat of Methane: Phys. Rev., vol. 37, 1931, p. 830.

14 Storch, H. H., The Free Energy and Entropy of Methane: Jour. Am. Chem. Soc., vol. 53, 1931, p. 1266. gave the value 43.39 entropy units, but an error in calculation of 0.52 units has subsequently been discovered.

15 Villars, D. S., The Entropy of Polyatomic Molecules: Jour. Am. Chem. Soc., vol. 53, 1931, p. 2006.

16 Rossini, F. D., The Heats of Combustion of Methane and Carbon Monoxide: Bur. Standards Jour. Research, vol. 6, 1931, p. 37.

17 See footnote 14.

18 Glockler, G., The Ionization Potential of Methane: Jour. Am. Chem. Soc., vol. 48, 1926, p. 2021.

are probably inside of an outer shell of 8 electrons. Thus the ionization potential of methane to form CH_4^+ is 14.5 volts and to form CH_3^+ is 15.5 volts,[19] these figures being of the order of magnitude of those for argon and krypton. Oxygen atoms were found by Harteck and Kopsch[20] to react quite rapidly with most organic compounds excepting methane where only a slow reaction was observed. Similarly Bonhoeffer and Harteck,[21] and von Wartenberg and Schultze[22] found that atomic hydrogen would not react with CH_4 to yield any product other than H_2 and CH_4. Ultra-violet light as obtained from a mercury arc lamp does not affect methane,[23] whereas ethane yields H_2, CH_4, C_4H_{10}, C_6H_{14} and other products. In general it may be concluded that the CH_4 molecule is quite resistant to chemical change, necessitating a large energy increment for activation at low temperatures.

THERMAL DECOMPOSITION

A review and bibliography of the thermal decomposition of hydrocarbons, and particularly of methane has been recently published by Egloff, Schaad, and Lowry,[24] and therefore only the more important and more recent data will be presented here.

In static experiments, in which CH_4 was confined in a sealed bulb of quartz or porcelain, it has been found that the decomposition at about 1,000° C. proceeds rapidly at first to carbon and hydrogen, but that the rate decreases practically to zero long before the equilibrium value is reached. Holliday and Exell[25] and Wheeler and Wood[26] postulate that the absorption by the walls of the reaction vessel of the hydrogen produced by the decomposition is responsible for the marked decrease in rate.

A recent paper by Holliday and Gooderham[27] shows conclusively that variation of the surface within wide limits has no measurable effect on the rate of decomposition of methane in quartz bulbs in the temperature range 900 to 1,100°C. These authors suggest that the initial reaction is a homogeneous bimolecular reaction, namely:

$$2\ CH_4 = C_2H_2 + 3\ H_2 \quad - 91{,}000 \text{ calories},$$

and that this reaction is followed by the decomposition of the acetylene at the surface of the reaction vessel. The apparent equilibria reached after the initially rapid reaction is completed are explained by them as a true equilibrium in the reaction given above for the formation of acetylene from methane; the rate of decomposition of the acetylene being decreased practically to zero due to the adsorption of hydrogen by the surface of the reaction vessel.

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19 Hogness, T. R., and Kvalves, H. M., The Ionization Process in Methane Interpreted by the Mass Spectrograph: Phys. Rev., vol. 32, 1928, p. 943.

20 Harteck, P., and Kopsch, U., Gas Reaktionen mit atomarem Sauerstoffe, Ztschr. phys. Chem., vol. B 12, 1931, p. 327.

21 Bonhoeffer, K. F., and Harteck, P., Über die Reaktionen von atomaren Wasserstoff mit Kohlenwasserstoffen: Ztschr. physik. Chem., vol. 139, 1928, p. 64.

22 von Wartenberg, H., and Schultze, G., Die Einwirkung atomaren Wasserstoffes auf Kohlenwasserstoffe, Ztschr. physik. Chem., vol. B 2, 1929, p. 1.

23 Kemula, W., Action of Ultra-Violet Rays on Aliphatic Hydrocarbons: Chem. Abs., vol. 24, 1930, p. 4462.

24 Egloff, G., Schaad, R. E., and Lowry C. D., jr., The Decomposition of the Paraffin Hydrocarbons: Jour. Phys Chem., vol. 34, 1930, p. 1617.

25 Holliday, G. C., and Exell, H. C., The Thermal Decomposition of Methane. Part I. Decomposition in Silica Bulbs: Jour. Chem. Soc. (London), 1929, p. 1066.

26 Wheeler, R. V., and Wood, W. L., Recent Experiments on the Pyrolysis of Methane: Fuel in Sci. and Practice, vol 9, 1930, p. 567.

27 Holliday, G. C., and Gooderham, W. J., The Thermal Decomposition of Methane. Part II - The Homogeneous Reaction: Jour. Chem. Soc. (London), 1931, p. 1594.

The mechanism suggested by Holliday and Gooderham is seen to be highly improbable, if one calculates the theoretical rate of the reaction (requiring at least 91,000 calories for activation) assuming that every collision between two methane molecules, which can supply the necessary energy of activation, is effective. Such a calculation (made by L. S. Kassel, associate physical chemist, Pittsburgh Experiment Station, U. S. Bureau of Mines) shows that the theoretical rate is approximately 10^5 times smaller than the actual rate.

In dynamic experiments varying yields of acetylene, ethylene, benzol, tar, and carbon are reported, the most important variables being the time of contact within the hot zone and the temperature. The character of the walls of the reaction tube exerts little or no influence on the reaction, provided strongly "methanizing" metals such as iron, nickel, cobalt, etc., are avoided. The latter catalyze the production of carbon and hydrogen. Quartz, porcelain, and copper are the materials commonly employed. Wheeler and Wood[28] find that dilution with nitrogen had only a slight effect on the yield of aromatic hydrocarbons at 1,000° C. using relatively long times of contact (1 to 2 minutes) in a quartz tube, whereas dilution with hydrogen results in markedly decreased yields. In a later paper[29] the same authors working at 1,050° C. in quartz tubes conclude that the important variable is the time of contact, rather than the surface exposed to the CH_4. This is illustrated by the data of Table 1 which also includes some data on the use of a chrome-iron reaction tube.

Table 1.- Wheeler and Wood's data on relative importance of time of contact and surface to volume ratio at 1,050° C. using 96 per cent of CH_4

Material	Surface to volume ratio	Time of contact, seconds	CH_4 decomposed to $C + 2H_2$, per cent	CH_4 converted to liquid products, per cent	CH_4 converted to higher hydrocarbons (mainly two carbon atoms), per cent
Quartz	1.86	11.2	45.5	8.1	10.3
		6.2	39.3	12.2	11.9
		4.3	32.6	15.0	16.6
		3.3	27.5	18.7	19.0
		2.6	22.6	18.2	24.3
Quartz	27.00	3.7	28.15	13.0	13.6
		1.6	18.21	15.2	22.0
		0.6	6.08	25.6	45.0
		0.45	4.06	15.0	61.3
		0.3	2.48	12.7	71.5
Chrome-iron	1.6	9.0	43.1	6.7	9.1
		4.5	33.5	9.8	13.7
		3.25	27.1	12.0	20.0
		2.5	18.1	12.1	31.9
		2.0	11.8	10.8	46.5

28 Wheeler, R. V., and Wood, W. L., The Pyrolysis of Methane: Fuel in Sci. and Practice, vol. 7, 1928, p. 535.
29 See footnote 26.

Wheeler and Wood found little or no acetylene or propylene in the decomposition products but butylene and butadiene were present in appreciable amounts.

The results of Fischer, et al,[30] are in substantial agreement with those of Wheeler and Wood. Stanley and Nash[31] report their maximum yields (using quartz tubes at 1,100° C., pure CH_4 at atmospheric pressure) of light oil, and tar were obtained with about 1 second contact time. Their off-gases contained about 1 per cent acetylene and about the same percentage of ethylene. Fischer and Meyer[32] report 5.9 per cent of C_2H_2 in their off-gases when using 760-millimeters pressure, 1,200 to 1,550° C., and 0.02 to 0.0003 second contact time with an electrically heated tungsten wire. At 50-millimeters pressure and 1,300 to 3,000° C., 9 to 15 per cent of C_2H_2 was obtained at 0.0005 to 0.0001 second contact time. The tungsten wire deteriorated rapidly, because of carbon deposition. Unfortunately, no details concerning the other constituents of the off-gas are given.

Frolich, White, and Dayton[33] working at about 1,100° C. and 30 to 40 millimeters of Hg pressure, obtained a maximum C_2H_2 content of about 6.5 per cent in the off-gases, the time of contact being about 0.05 seconds. The off-gas containing 6.5 per cent of C_2H_2 also contained about 6 per cent of H_2. Theoretically the H_2 content should have been at least three times that of C_2H_2. It is possible that some H_2 may have been lost by diffusion through the quartz tube. The authors also find that the C_2H_2 yield is very sensitive to the time of contact and the pressure. The percentage of C_2H_2 found did not vary much with either time of contact or pressure, but remained approximately constant at 1.5 to 2.0 per cent. The amount of carbon, tar, and oil formed decreased rapidly with decrease in pressure and time of contact.

De Rudder and Biedermann[34] have done a most careful and complete research in so far as the production of acetylene by the pyrolysis of methane is concerned. They performed their experiments in quartz, at temperatures of 900 to 1,500° C. and pressures of 760 millimeters to 34 millimeters of Hg, using 99.7 per cent of CH_4, 0.3 per cent of N_2 as raw material. Their results are of sufficient importance to reproduce here in Table 2. Only very small amounts of tar and oil were obtained at 1,000° C. Above this temperature little or no tar or oil was observed.

The results of Table 2 show that the maximum C_2H_2 content was obtained at 1,200° C. using 100-millimeters pressure and 2.0 seconds contact time; and the maximum C_2H_2 content at 1,500° C., 44 millimeters pressure, and 0.013 second contact time. In the latter experiment 52.5 per cent of the CH_4 was converted mainly to C_2H_2 and only 10.5 per cent to carbon, 37 per cent being recovered unchanged.

The results given in Table 2 indicate that ethylene is the primary product of the thermal decomposition, rather than acetylene. Thus at 1,000°C. little if any acetylene is present, whereas the ethylene content is of the order of 1 to 2 per cent, and is increasing with decreasing time of contact. Similarly at 1,300° C. the ethylene content increases from 0.6 per cent to 1.7 per cent when the time of contact is changed from 0.51 to 0.05 second, respectively, whereas the acetylene content decreases from 6.6 per cent to 2.7 per cent with the same change of contact time.

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30 Fisher, F., Pichler, H., Meyer, K., and Koch, H., The Formation of Benzol and other Hydrocarbons by the Action of Heat on Methane: Brenn Chem., vol. 9, 1928, p. 309; 2nd. Internat. Conference on Bit. Coal. vol. 2, 1928, p. 789.

31 Stanley, H. M., and Nash, A. W., The Production of Gaseous, Liquid and Solid Hydrocarbons from Methane. Part I - The Thermal Decomposition of Methane: Jour. Soc. Chem. Ind., vol. 48, 1929, p. 1, T.

32 Fisher, F., and Meyer, K., Über die thermische Bildung von Acetylen aus Methan: Brenn. Chem., vol. 10, 1929, p.324.

33 Frolich, P. K., White, A., Dayton, H. P., Studies on Production of Acetylene from Methane. I - Cracking Under Vacuum: Jour. Ind. Eng. Chem., vol. 22, 1930, p. 20.

34 Rudder, Fr. de, and Biedermann, H., Sur la pyrogenation du Methane: Compt Rend., vol. 190, 1930, p. 1194; Bull. Soc. Chim., vol. 47, 1930, p. 710.

Table 2.- Results of Rudder and Biedermann's experiments on thermal decomposition of methane

Temp., °C.	Time of contact, seconds	Pressure, millimeters of Hg	Per cent in off-gases of CH_4	H_2	C_2H_2	C_2H_4	Per cent conversion of CH_4 to C	C_2H_2	Per cent of CH_4 undecomposed
900	75.0	760	84.8	8.9	0.2	1.6	3.0	4.0	93.0
1000	0.7	87	-	3.8	-	0.2	3.5	0.5	96.0
1000	4.9	760	83.4	14.0	0.5	1.7	5.0	4.0	91.0
1000	15.8	760	42.1	55.1	-	1.8	36.0	5.0	59.0
1000	58.0	760	34.7	64.0	-	0.9	46.0	2.0	52.0
1200	0.36	53	-	6.55	-	0.9	2.5	2.5	95.0
1200	0.8	81	-	13.0	-	2.2	4.5	4.5	91.0
1200	2.0	105	-	29.0	2.3	3.4	9.0	14.0	77.0
1300	0.05	151	64.4	27.4	2.7	1.7	9.0	11.0	80.0
1300	0.14	118	33.6	57.1	6.0	0.9	28.5	21.0	50.5
1300	0.51	142	13.8	76.1	6.6	0.6	49.0	26.0	25.0
1400	0.043	134	17.8	73.6	5.6	0.1	48.0	20.0	32.0
1400	0.10	82	21.7	55.9	11.0	1.5	17.5	44.0	38.5
1400	0.13	34	21.9	63.5	10.6	0.9	25.0	38.0	37.0
1500	0.007	60	52.7	34.2	6.7	3.3	4.5	26.0	70.6
1500	0.013	44	21.5	59.4	14.85	1.3	10.5	52.5	37.0
1500	0.2	73	2.2	30.6	14.4	-	37.0	58.2	4.54

The work of Hague and Wheeler[35] also indicates that ethylene is the parent substance for the formation of benzol; the intermediary stages being butylene, butadiene, and hexadiene. Naphthalene may be formed by way of the condensation of butadiene with benzol, and anthracene from naphthalene and butadiene.

Wheeler[36] subjected the work of Bone and Coward,[37] Wheeler and Wood,[38] and Holliday and Exell[39] to an analysis based on the assumption that in both the static and streaming experiments the reaction was entirely heterogeneous, hydrogen exercising a marked retarding influence. The curves given by Bone and Coward and Wheeler and Wood indicate that in static experiments after the first rapid decomposition has ceased, a reaction of zero order (amount of methane decomposed per unit time is constant and independent of its pressure) becomes apparent. Holliday and Exell's results, however, do not supply any evidence for such zero order reaction. Wheeler develops rate equations of the type:

$$- dp/dt = k_1p^n \quad (1 - k_2 \{(P - p)^a /p\}^b),$$

where p is the pressure of CH_4 at time t, and is calculated from the initial pressure P and the observed total pressure 2 P - p at time t. p indicates the order of the reaction as

35 Hague, E. N., and Wheeler, R. V., The Mechanism of Thermal Decomposition of the Normal Paraffins: Jour. Chem. Soc. (London), 1929, p. 378.

36 Wheeler, T. S., The Kinetics of the Thermal Decomposition of Methane: Fuel, vol. 10, 1931, p. 175.

37 Bone, W. A., and Coward, H. F., The Thermal Decomposition of Hydrocarbons: Jour. Chem. Soc., vol. 93, 1908, p.1197.

38 See footnotes 26 and 28.

39 See footnote 25.

regards CH_4. k_1 is the velocity constant for the particular reaction vessel used, and $k_2 ((P - p)^a/p)^b$ represents the retarding influence of hydrogen. In all cases Wheeler puts n = 1, and for Holliday and Exell's results a = 2, b = 0.25, whereas for the other investigators a = 3, b = 1/9. While these equations represent the results with fair accuracy, the author (Wheeler) states that no physicochemical interpretation of them can be given at present. Heats of activation between 30,000 and 34,000 calories are calculated by Wheeler.

This complexity of the retardation of CH_4 decomposition by hydrogen is not without parallel, for Hinshelwood[40] cites a similar case—namely, the decomposition of NH_3 on quartz. He states:

> In the equation $dx/dt = (k(a - x))/(1 + bx)$, the factor $1/(1 + bx)$, which represents the retarding effect, reduces as we have seen to $1/bx$ for large values of b and becomes equal to unity when b vanishes. For intermediate conditions it may be represented approximately by an inverse fractional power of x. This is the origin of the relation sometimes found empirically that the retarding effect of a gas is proportional to the square root of its pressure.

It will be observed that Wheeler's equation when n = 1, a = 2, and b = 0.25, presents just this proportionality of the retarding effect.

The writers chief criticism of Wheeler's paper concerns the assumption that in the streaming experiments the reaction is entirely heterogeneous. If this were the case, it would be very difficult to account for the observation by several experimenters that a large increase in the surface exposed to the CH_4 does not alter the reaction rate appreciably. This indicates that while the reaction may be initiated at the walls, it is probably largely homogeneous after it is once started. A chain reaction with long chains is also ruled out, for in this case the rate would be decreased by increased surface exposure.

The primary steps in the thermal decomposition of methane are as yet quite obscure. The following series of reactions should, however, be considered in any experiments on the primary steps:

$$CH_4 + \text{hot quartz or hot carbon} \rightarrow C + 2H + H_2 \quad (1)$$
$$CH_4 \text{ (adsorbed)} + H \text{ (adsorbed)} \rightarrow CH_3 + H_2 \quad (2)$$
$$CH_3 + CH_4 \rightarrow C_2H_5 + H_2 - 11{,}000 \text{ cals.} \quad (3)$$
$$C_2H_5 \rightarrow C_2H_4 + H - 20{,}000 \text{ cals.} \quad (4)$$
$$CH_3 \rightarrow CH_2 + H - 25{,}000 \text{ cals.} \quad (5)$$
$$CH_2 + CH_4 \rightarrow C_2H_4 + H_2 + 20{,}000 \text{ cals.} \quad (6)$$
$$CH_4 + H \rightarrow CH_3 + H_2 - 25{,}000 \text{ cals.} \quad (7)$$

The first step is the complete dissociation of methane to form a layer of carbon atoms on the surface of the reaction vessel. The hydrogen resulting from the decomposition may be partially adsorbed as monatomic hydrogen on the carbon atoms, and the remainder as molecular hydrogen. The second step consists in the reaction of adsorbed methane with adsorbed monatomic hydrogen to form a methyl radical and molecular hydrogen. The heat of formation of molecular hydrogen is somewhat less than that necessary to break an $H_3C - H$ bond, and hence the reaction is only mildly exothermic. Some energy of activation may be necessary to activate the methane.

It is probable that as soon as atomic hydrogen is present due to reactions (4) and (5) the synthesis of ethylene will be a homogeneous chain reaction. The chains will, however, be very short, due to the continuous removal of atomic hydrogen by the walls of the reaction

40 Hinshelwood, C. N., Kinetics of Chemical Change in Gaseous Systems: Oxford, 1929, 2nd ed., p. 209.

vessel. The effect of increased surface exposure can not be readily predicted, for while the rates of reactions (1) and (2) would be increased, the rate of reactions (7) would be decreased because of the recombination of hydrogen atoms at the surface to form molecular hydrogen.

The heats of reaction listed in reactions (3) to (7), inclusive, are approximations based on the estimates of Mecke[41]. The probability of reaction (4) has recently been discussed by Rice.[42]

EFFECT OF ELECTRIC DISCHARGE THROUGH METHANE

Egloff, Shaad, and Lowry[43] present a comprehensive review of research on the electrical decomposition of methane up to early in 1930. At that time the most interesting results from a practical point of view were those of Fischer and Peters[44] who were able to obtain 82.3 per cent conversion of CH_4 to acetylene by passing coke-oven gas through an electrical discharge tube at about 50 millimeters pressure. The electrodes were of Krupp V 2A steel and about 50 centimeters apart, about 2,000-volts, 50-cycle current being employed. The energy required per cubic meter of acetylene was 31 kilowatt-hours. Subsequently Peters and Pranschke[45] showed that in passing 1160 liters per hour of a gas (containing 54.6 per cent of H_2, 24.9 per cent of CH_4, 9.7 per cent of N_2, 2.1 per cent of CO_2, and 0.9 per cent of O_2) through an electric discharge tube (5-centimeter diameter electrodes 30 centimeters apart) at 40 to 60 millimeters Hg pressure, the off-gas contained 8.0 per cent of C_2H_2; no carbon was deposited, and the energy consumption was 11.6 kilowatt-hours per cubic meter of acetylene produced. This corresponds to about 30 per cent conversion of the CH_4. At lower pressures (10 to 20 millimeters) a much higher conversion--namely, about 95 per cent of CH_4 to acetylene is claimed.

As reported in a later paper, Peters and Wagner,[46] using a gas containing 91.9 per cent of CH_4, 3.4 per cent of N_2, 3.5 per cent of H_2, and 0.9 per cent of (O_2 + CO) at about 90 millimeters pressure and 0.05 second contact time, obtained a minimum energy value of 13 kilowatt-hours per cubic meter of acetylene, the product containing 4.2 per cent of C_2H_2. Low concentrations (about 0.8 per cent) of ethylene were also obtained. Dilution with hydrogen favors the conversion of CH_4 to acetylene up to a maximum when the ratio of CH_4 to H_2 is 2 : 1. At lower pressures (that is, less than 10 millimeters) appreciable amounts of C_2H_6 and C_3H_3 appeared in the off-gases. The minimum energy consumption per cubic meter of acetylene produced for any given set of conditions (that is, space velocity and pressure particularly) is obtained when the power input is increased just to the point where the color of the discharge changes from blue to yellow. This change is associated with the appearance of a new carrier--namely, C_2 ions. It is not possible at this time to present a detailed picture of the mechanism of the production of acetylene from CH_4 by electric

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41 Mecke, R., Experimentelle Ergebnisse und Ziele der Bandenforschung: Ztschr. Elektrochem., vol. 36, 1930, p. 595.

42 Rice, F. O., The Thermal Decomposition of Organic Compounds from the Standpoint of Free Radicals. I - Saturated Hydrocarbons: Jour. Am. Chem. Soc., vol. 53, 1931, p. 1959.

43 See footnote 24.

44 Fischer, F., and Peters, K., Über die Einwirkung Elektrischen Entladungen auf Kohlenwasserstoffhaltige Gase bei Vermindertem Druck: Ztschr. physik. Chem., vol. A 141, 1929, p. 180.
Fischer, F., and Peters, K., Über die Umwandlung von Methan bzw. Koksofengas durch Elektriche Entladungen bei Unterdruck: Brenn. Chem., vol. 10, 1929, p. 108.

45 Peters, K., and Pranschke, A., Neue Versuche über die Umsetzung von Methan aus Koksofengas in Acetylen durch Elektrische Entladungen: Brenn. Chem., vol. 11, 1930, p. 239.

46 Peters, K., and Wagner, O. H., Athylen - und Acetylenbildung aus Methan in Elektrischen Entladungen: Ztschr. physik. Chem., vol. A 153, 1931, p. 161.

discharge, especially since Eisenhut and Conrad[47] have shown by mass spectrographic methods that every species of ion from one to four carbon atoms is present in the discharge.

In the recently published work of Brewer and Kueck[48] on the decomposition of methane in the glow discharge using much lower voltages (several hundred volts) and pressures (a few millimeters of mercury) than Fischer and his coworkers and immersing the discharge tube in liquid air, practically quantitative conversion of methane to ethylene was obtained. These authors suggest that the reaction is essentially the following:

$$(CH_4)^+ + CH_4 \rightarrow (C_2H_4)^+ + H_2 ,$$

the $(C_2H_4)^+$ being rapidly freed of its electric charge by neutralization at the walls of the discharge tube. The fact that in both the thermal decomposition of and electric discharge through CH_4, ethylene appears to be one of the earliest products is perhaps signigicant, but at present it can be regarded only as a coincidence with little or no bearing on the mechanism of the two systems of reaction.

OXIDATION OF METHANE

Egloff and Schaad[49] have reviewed the literature on the oxidation of methane up to 1929. The main objective of most of the research work on this subject has been to obtain an industrially feasible process for the production of formaldehyde. Since the production of cheap methanol from water gas, the search for such a process has practically ceased, because of the ease with which formaldehyde can be produced from methanol.

The kinetics of the oxidation of methane has been studied by Pease and Chesebro[50] and more recently by Fort and Hinshelwood[51]. The oxidation is apparently a chain reaction, for the rate is decreased by an increase in the surface exposed to the reactants.

The relation between the rate and concentrations of methane and oxygen is of no simple order, the nearest integral value being 3 or 4. The rate increases rapidly with increasing CH_4 concentration but is relatively little influenced by the O_2 concentration. An induction period exists in the oxidation of CH_4, during which the rate of reaction is low. Fort and Hinshelwood state that this induction period probably represents the time necessary for the production of the "steady state" concentration of formaldehyde, for a similar induction period exists in the oxidation of methanol.

The partial oxidation of methane using $2CH_4 + 1\ O_2$ and producing a mixture of water gas and acetylene was studied by Fischer and Pichler[52]. The results of this work are given in the following table.

The mixture of water gas and acetylene as obtained in the experiments of Table 3 would probably give a considerably higher yield of gasoline when passed over proper catalysts at about 200° C., than is obtained with water gas alone.

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47 Eisenhut, O., and Conrad, R., Beobachtungen ber Zerfall und Bildung von Kohlenwasserstoffen in Entladungsröhren mit hilfe von Kanalstrahlen: Ztschr. Elektrochem, vol. 36, 1930, p. 654.

48 Brewer, A. K., and Kueck, P. D., Chemical Action in the Glow Discharge. VII -. Dissociation and Oxidation of Methane: Jour. Phys. Chem., vol. 35, 1931, p. 1293.

49 Egloff, G., and Schaad, R. E., The Oxidation of the Paraffin Hydrocarbons: Chem. Rev , vol. 6, 1929, p. 91.

50 Pease, R. N., and Chesebro, P. R., Characteristics of Homogeneous Exothermic Gas Reactions: Proc. Nat. Acad. Sci., vol. 14, 1928, p. 472.

51 Fort, R., and Hinshelwood, C. N., Further Investigations on the Kinetics of Gaseous Reactions: Proc. Roy. Soc., vol. A, 129, 1930, p. 284.

52 Fischer, F., and Picnler, H., Über die Partielle Verbrenung von Methan bei Verschiedenen Drucken, mit bosondere Berucksichtigung der hierbei auftretenden Acetylenbildung: Brenn. Chem., vol. 11, 1930, p. 501.

Table 3.- Partial oxidation of methane by oxygen

2.5 by 100 millimeter Pythagorasmasse tube 105 liters per hour of $2CH_4 + 1\ O_2$ for 5 hours at 1,150 to 1,200° C.

	Volume, per cent							
	CO_2	C_2H_2	O_2	CO	H_2	CH_4	C_2H_6	N_2
Reactants	0	0	33.6	1.8	0.4	59.4	0.0	4.8
Products	2.5	9.5	0.2	23.5	50.9	10.0	0.0	3.4
	Also 20 c. c. of "light oil."							
$CH_4 + O_2$ passed through 2.5 by 100 millimeter tube at 1,400° C.								
O_2 liters/hour	0	20	30	40	50	60	70	80
$O_2 + CH_4$ liters/hour	109	136	150	156	174	183	204	234
CO_2, volume, per cent	0.2	0.9	1.6	1.4	2.4	3.0	3.0	3.4
C_2H_2 do	5.2	7.6	8.2	9.0	9.4	9.4	6.6	1.2
O_2 do	0.3	0.5	0.6	0.2	0.0	0.0	0.0	0.2
CO do	1.2	12.4	16.3	19.6	22.7	25.8	30.4	35.8
H_2 do	25.5	38.6	44.5	48.5	53.1	54.2	56.3	56.4
CH_4 do	64.3	37.0	25.8	18.6	9.4	4.8	1.9	0.8
CH_4 to C_2H_2 do	12.2	21.4	26.6	30.4	34.8	37 0	29.4	5.0

Bone[53] has recently found that the direct oxidation of methane, using a mixture of $9\ CH_4 + 1\ O_2$ at 360° C. and 100 atmospheres pressure, yielded considerable quantities of (17 per cent of the CH_4 reacting to form) methanol, the bulk of the products being CO, CO_2 and H_2O.

REACTION OF METHANE WITH STEAM AND CARBON DIOXIDE TO PRODUCE WATER GAS

The production of water gas by way of the hydrolysis of methane is an exothermic reaction which requires a temperature of about 1,000° C. for practically complete conversion of the methane. About the same temperature is essential for the reaction with carbon dioxide. Fisher and Tropsch[54] investigated the activity of a variety of materials as catalysts for the CO_2 reaction. Copper, iron, Mo, and W were found to have little or no catalytic activity at 850° C., whereas CO or Ni promoted with Al_2O_3 is an excellent catalyst. Hawk, Golden, Storch, and Fieldner[55] found Co to be a much poorer catalyst than Ni for the H_2O reaction They obtained high conversions using a Ni catalyst prepared by impregnating an Al_2O_3 + clay refractory (corundite) with $Ni(NO_3)_2$ solution and subsequently reducing to metallic nickel. Their process consists of alternately blasting the catalyst-filled converter with $CH_4 + O_2$ and then $CH_4 + H_2O$. Purging after the heating blast is of course essential to remove N_2, CO_2, etc.

53 Bone, W. A , Formation of Methyl Alcohol by Direct Oxidation of Methane: Nature, vol. 127, 1931, p. 481.

54 Fischer, F., and Tropsch, H , Die Umwandlung von Methan in Wasserstoff und Kohlenoxid. Brenn. Chem. vol. 9, 1928, p. 39.

55 Hawk, C. O., Golden, P., Storch, H. H., and Fieldner, A C., The Conversion of Methane to Carbon Monoxide and Hydrogen: Ind. and Eng. Chem To be published.

Gluud, Klempt, Brodkort, et al.,[56] found it possible to use an alloy tube (20 per cent of Ni, 25 per cent of Cr, 55 per cent of Fe) and external heating, on the catalytic (Ni on burned dolomite or on Al_2O_3) hydrolysis of methane. There was, however, a slow, but continuous loss in weight by the alloy tube.

Peters and Pranschke[57] studied the reaction of methane with H_2O and CO_2 in the electric discharge. At 50 millimeters pressure they obtained quantitative conversion to $CO + H_2$ when the current density was sufficiently high. At lower current densities varying quantities of acetylene were obtained. Their most interesting result was the conversion of a mixture of $1CO_2 + 2CH_4$ into a gas which contained 26.1 per cent of CO, 53.3 per cent of H_2, 10.0 per cent of C_2H_2, 4.7 per cent of CO_2, 3.6 per cent of CH_4, and 1.04 per cent of higher hydrocarbons. Such a product would be excellent raw material for gasoline production.

The catalytic hydrolysis of methane at high pressures and at 250 to 500° C. is claimed by Dreyfuss[58] to yield methanol and higher alcohols.

REDUCTION OF METALLIC OXIDES BY METHANE

The use of methane (or natural gas) for metallurgical purposes, has received comparatively little attention. Bouton[59] studied the reduction of hematite to magnetite by methane-hydrogen mixtures and found the methane reaction to be slow. Thus even at 900° C. the time of contact required for 90 per cent utilization of the methane was 34 seconds. Thermodynamic calculations made by the writer indicate that it may be possible to reduce either hematite or magnetite to metallic iron by methane without carbon deposition if a reaction temperature above 650° C. is employed. Below this temperature carbon deposition is practically certain to occur, whereas above it some temperature may be found at which a sufficiently rapid rate of reduction combined with little or no carbon deposition is obtained. Preliminary experiments by the Bureau of Mines indicate that about 900° C. is essential for a sufficiently rapid reaction rate, and that if methane free from higher hydrocarbons is used, little or no carbon deposition occurs. The reductions of both hematite and magnetite are endothermic, and 1,500 to 2,000 B.t.u. (depending upon the character of the ore and the temperature of reduction) must be supplied per pound of metallic iron produced.

The reduction of zinc oxide by methane was subjected to a thermodynamic analysis by C. G. Maier[60] who found 870° C. to be the minimum temperature necessary to avoid carbon deposition. Subsequent experimental work by Doerner[61] showed that in the temperature range 975 to 1,000° C. little or no carbon is deposited and the rate of reduction of zinc oxide and vaporization of zinc metal is sufficiently rapid to compare favorably with that obtained from retorts by standard practice. The metal produced is of exceptional purity.

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56 Gluud, G., Klempt, W., Brodkort, F., et al., Ber. Ges. für Kohlentechnik mb.H. 'Wasserstoff Heft". vol. 3, 1930, pp. 211-370.

57 Peters, K., and Pranscke, A., Die Umsetzung des Methans mit Kohlendioxid und Wasserdampf in Elektrischen Entladungen: Brenn. Chem., vol. 11, 1930, p. 473.

58 Dreyfuss, -.: French Patent 698,094, June 26, 1930.

59 Bouton, C. M., The Rate of Reduction of Hematite to Magnetite by Methane: Rept. of Investigations 2381, Bureau of Mines, 1922, 9 pp.

60 Maier, C. G., Zinc Smelting from a Chemical and Thermodynamic Viewpoint: Bull. 324, Bureau of Mines, 1930, 93 pp.

61 Doerner, H. A., Reduction of Zinc Oxide by Methane or Natural Gas: Rept. of Investigations 3091, Bureau of Mines, 1931, 14 pp.

Experiments on the reduction of copper and cobalt oxides by methane are reported by Campbell and Gray.[62] Carbon dioxide and water are the chief products in the temperature range studied—namely, 300 to 650° C.

SYNTHESIS OF HYDROGEN CYANIDE FROM METHANE AND AMMONIA

Hydrogen cyanide has for a long period of years been produced by the treatment of sodium cyanide with sulphuric acid. In recent years the synthesis of formamide from CO and NH_3 has led to experimentation on the catalytic dehydration of formamide to produce HCN. Research on the synthesis of HCN directly from CO and NH_3 has also received the attention of Bredig, Elöd, at al.[63,64,65] The best yield obtained by these experimenters was when using a twentyfold excess of CO at 700° C. and a cerium oxide-alumina catalyst; 65 per cent conversion of NH_3 to HCN with 14 per cent loss of NH_3 by decomposition. The synthesis of HCN from hydrocarbons and ammonia by the use of contact catalyst was investigated by Bredig, Elöd, and Demme.[66] The yields with equimolar mixtures of NH_3 with CH_4, C_2H_2, and C_2H_4 were as follows:

Table 4.- Formation of HCN from hydrocarbons

Reactants	Catalyst	Temp., °C.	NH_3 converted to HCN, per cent	NH_3 lost by decomposition, per cent
$NH_3 + C_2H_4$	$SiO_2 + Al_2O_3$	800	71.5	19.5
$NH_3 + C_2H_2$	Al_2O_3	800	40.0	23.6
$NH_3 + CH_4$	Al_2O_3	1000	10.7	89.3

The reaction of methane with ammonia to form HCN appears to proceed by way of acetylene or ethylene for no appreciable quantities of HCN are formed until temperatures close to 1,000° C. are employed.

Fischer and Peters[67] found that when an excess of nitrogen is added to CH_4 and the mixture passed through an electric discharge (about 5,000 volts for 40 centimeters electrode distance), the methane is practically completely converted to HCN. Subsequently Peters and Küster[68] studied the reaction between methane and ammonia in the electric discharge. They used a 5 by 75 centimeter tube containing electrodes of Krupp V2A steel 40 centimeters apart.

62 Campbell, J. R., and Gray, T., The Influence of various Catalysts in Promoting the Oxidation of Methane by Means of Copper Oxide: Jour. Soc. Chem. Ind., vol. 49, 1930, p. 447.

63 Bredig, G., Elöd, E., and Müller, Rudolf K., Zur Kenntnis der Katalytischen Blausaurebildung. Part III - Blausaurebildung aus Kohlenoxyd und Ammoniak: Ztschr. Elektrochem., vol. 36, 1930, p. 1003.

64 Bredig, G., Elöd, E., and Kortüm, G., Zur Kenntnis der Katalytischen Blausaurebildung. Part IV. Blausaurebildung aus Kohlenoxyd und Ammoniak: Ztschr. Electrochem., vol. 36, 1931, p. 1007.

65 Bredig, G., Elöd, E., and König, Walter, Zur Kenntnis der Katalytischen Blausaurebildung. Part V - Über Ceroxyd als Katalysator bei der Blausaurebildung: Ztschr. Elektrochem., vol. 37, 1931, p. 2.

66 Bredig, G., Elöd, E., and Demme, Ernst, Zur Kenntnis der Katalytischen Blausaurebildung. Part II - Blausaurebildung aus Kohlenwasserstoffen und Ammoniak: Ztschr. Elektrochem, vol. 36, 1930, p. 991.

67 See footnote 44.

68 Peters, K., and Küster, H., Blausaurebildung in Elektrischen Entladungen: Brenn. Chem., vol. 12, 1931, p. 122.

One phase of 220-volt 50-cycle current was fed to a 220/10,000 volt transformer to furnish high-voltage current. They found that at 120 liters per hour of flow of $1CH_4 + 1NH_3$ at 20 to 40 millimeters pressure and an energy consumption of 2.0 kilowatts, about 70 per cent of the CH_4 was converted to HCN with 23.5 per cent loss of NH_3. In this experiment the highest power input investigated by them was employed, and it is probable that higher conversions can be obtained with higher current densities. The following table shows their results on the influence of the ratio of CH_4 to NH_3, the experiments being conducted in the apparatus described above, and the gas flow being about 120 liters per hour in each case.

Table 5.- Influence of CH_4/NH_3 ratio on HCN production by electric discharge

Experiment No.	CH_4, per cent	NH_3, per cent	Voltage	Pressure, mm.Hg	CH_4 converted to HCN, per cent	NH_3 converted to HCN, per cent	CH_4 converted to C_2H_2, per cent	NH_3 decomposed, per cent	Higher hydrocarbons in off gas,[1] per cent
11	30	70	6460	35	90.3	38.7	3.9	62.4	0.3
5	50	50	5820	30	66.6	66.7	18.1	23.4	3.2
7	50	50	5640	23	69.7	69.7	17.1	23.5	2.7
9	70	30	5820	27	38.0	88.9	48.0	9.4	9.6
10	85	15	5640	28	22.2	100.0	58.7	0.0	12.8

1 Mainly acetylene.

Experiment No. 10 is of special interest, for combined with complete conversion of NH_3 to HCN, 58.7 per cent of the CH_4 is obtained as acetylene.

HALOGENATION OF METHANE

The data on the reactions of methane and of other hydrocarbons with the halogens has been very recently summarized by Egloff, Schaad, and Lowry[69] so that it is unnecessary to review such literature here. Our knowledge of the chemistry of the halogenation of methane is in a much more satisfactory state than any other of its reactions, although there is ample opportunity for further research, as indicated by Egloff, et al.

SUMMARY

A brief critical review of the more important data on the physical and chemical properties of methane is presented, the primary object being to provide an introduction to further research on the utilization of waste natural gas. Perhaps the most interesting section of this review concerns the thermal decomposition of methane. Some suggestions concerning the mechanism of this decomposition are presented. The first section of the paper presents the thermodynamic properties of methane, and subsequent sections discuss the thermal and electrical decompositions, the oxidation by oxygen, carbon dioxide, and metal oxides, the hydrolysis by steam, and the synthesis of hydrogen cyanide in the electric discharge from methane and nitrogen.

69 Egloff, G., Schaad, R. E., and Lowry, C. O., jr., Halogenation of the Paraffin Hydrocarbons: Chem. Rev., vol. 8, 1931, p. 1.

I. C. 6550

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Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MILLING METHODS AND COSTS AT THE SUPERIOR CONCENTRATOR OF THE ENGELS COPPER MINING CO., PLUMAS COUNTY, CALIF.

BY

W. I. Nelson

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MILLING METHODS AND COSTS AT THE SUPERIOR CONCENTRATOR OF THE ENGELS COPPER MINING CO., PLUMAS COUNTY, CALIF.[1]

By W. I. Nelson[2]

INTRODUCTION

This paper, which describes the milling practice of the Engels Copper Mining Co., is one of a series on milling methods being prepared for and published by the United States Bureau of Mines.

ACKNOWLEDGMENTS

Acknowledgment is made of the assistance rendered by J. Casagrande, mill foreman of the Engels concentrator, in collecting much of the data presented in this paper.

GENERAL DESCRIPTION

The mill of the Engels Copper Mining Co. is located in Lights Canyon, Plumas County, Calif., 22 miles by railroad from Paxton on the Western Pacific Railroad. The mill treats ore from two mines, the Engels and the Superior, in the proportion of 8 : 2, respectively. The Engels ore is mined entirely by the shrinkage-stope system, the ore being broken to pass 11-inch spaces between grizzly rails. In the Superior mine the pitch of the veins varies from 35 to 85° from the horizontal. The steep veins are mined by shrinkage stopes and the flat veins by the room and pillar system. No timber is used to support the walls in either mine. The ore from the Engels mine is delivered to a concrete mill bin in 6-ton capacity cars, which are dumped by a revolving tipple. The Superior ore is dumped into a steel mill bin by 5-ton capacity skips operated in a vertical shaft.

The concentrator can treat 1,500 tons of ore per day, but averages only 1,000 tons per day owing to the limited capacity of the mines.

Water for milling purposes flows to the concentrator by gravity except during exceptionally cold spells in winter and the dry months of fall; at such periods additional water is pumped from Lights Creek, a lift of 350 feet.

Electric power is purchased from the Great Western Power Co. and this supply is supplemented during the spring and winter months by 400 hp. generated locally by water power.

ORE TREATED

The copper minerals in the ore are principally chalcopyrite and bornite in the average proportion of 3 : 2, respectively. During 1929 the heads averaged 1.524 per cent of copper and from concentrates returns were estimated to contain $0.11 in gold and 0.345 ounce of silver per ton. In the Superior mine, the vein gangue is chiefly a fine-grained tourmaline

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from United States Bureau of Mines Information Circular 6550."

2 - General superintendent, Engels Copper Mining Co., and consulting engineer, United States Bureau of Mines.

which occurs in a hard quartz monzonite. The chalcopyrite and bornite occur in widely varying proportions scattered through the vein matter. Several veins carry small amounts of tetrahedrite, usually high in silver content, and minor amounts of galena and sphalerite. In the Engels mine the copper minerals occur in altered diorite; the chalcopyrite and bornite are much more intimately mixed than is the case at the Superior mine and occur in an interwoven network of stringers and in fine-grained disseminations through the rock. Magnetite generally occurs through the ore zone in sufficient quantities to cause pieces of ore to be lifted when brought into a powerful magnetic field. Pyrite is present in very small amounts, but does not constitute a problem in flotation treatment.

On account of the fineness of the sulphide particles in the disseminated portions of the orebody it is necessary to grind the ore very fine in order to make a satisfactory extraction and fairly clean concentrates.

HISTORY OF CONCENTRATOR OPERATIONS

A small mill of 150 tons capacity per day was erected at the Upper Camp of the Engels mine in 1914. The capacity was increased later to 400 tons per day by the addition of another grinding unit. It is understood that this mill was one of the first, if not the first, in the United States to practice the all-flotation method of treatment. This plant treated practically all of the oxidized ore from the upper workings of the mine, and copper losses in the tailings were naturally high on this account. Chalcocite occurred in both the sooty and clean steel-gray mineral varieties, neither of which appeared to give any trouble in flotation operations.

In this mill the primary grinding was done by two ball mills; one 8-foot by 36-inch Hardinge, and one 5 by 6 foot Allis-Chalmers. The coarser sands contained in the pulps of the primary mills were further reduced by tube mills.

The ground pulp from the Hardinge unit was treated in four Callow cells which produced finished concentrates and tailings. The tailings were given additional flotation treatment in an 18-inch Minerals Separation machine which produced finished concentrates, middling froths, and finished tailings. The middlings were retreated in the cleaner cells of the Allis-Chalmers unit.

The pulp from the Allis-Chalmers unit was divided between one 12-inch and one 18-inch Minerals Separation machines. The 18-inch machine also handled the tailings of the primary Callow cells of the Hardinge unit. These machines produced concentrates, which were cleaned by a set of Callow cells, and final tailings. The cleaner cells produced finished concentrates and middlings which were returned to the tube mills for additional grinding.

Until 1917, when the Indian Valley Railroad was completed to Engels, the concentrates were trammed down the hill, a distance of 1 mile, and then hauled by truck to Keddie. The transportation of concentrates and supplies by truck during the winter months over dirt roads and mountain grades was a source of much trouble.

In 1918 a new concentrator was completed. This plant, located near the Superior mine and at an elevation which is 1,000 feet lower than the first mill, treated ores from both the Engels and Superior mines. The Engels ore was trammed from the mine to a jaw crusher located near the portal of the No. 6 tunnel. After crushing, the ore was transported to the new mill by an aerial tram.

The grinding unit as first operated in the new concentrator comprised two primary ball mills, one 8 by 6 foot Marcy and one 8-foot by 36-inch Hardinge, with two 6 by 12 foot tube mills for secondary grinding. Owing to the toughness of the ores from the Superior mine and from the deeper workings of the Engels mine the capacity of the Marcy mill operating on coarse feed was far from expectations. A set of rolls was, therefore, installed to reduce

the size of the feed to this mill. The rolls received as feed the oversize of a trommel having 2-inch holes, and after one pass through the rolls the ore was conveyed to the ball-mill feed bins.

As the mine production increased, two 8-foot by 36-inch Hardinge mills were added to increase primary grinding capacity. When the old mill was dismantled, one 7 by 10 foot ball-tube mill and one 8-foot by 36-inch Hardinge mill were moved to the new concentrator and installed as secondary grinding units. These changes were completed in 1920 and since that time only minor changes have been made in the primary and secondary grinding units. The 7 by 10 foot ball-tube mill was reduced to 6 by 10 foot size by lining the shell with 6 by 6 inch timbers and covering these with cast-iron liners. A model D classifier was moved from a primary Hardinge mill to the 6 by 10 foot tube mill in order to increase the capacity of the latter by operating it with a greater circulating load. The Hardinge mill, from which this classifier was moved, is now operated in circuit with a model C classifier. The model C machine handles the load in a satisfactory manner when operated in open circuit with the ball mill but must be watched closely when operating in closed circuit to guard against overloading. A classifier drainage sump was installed to facilitate the starting up of grinding units after power interruptions.

A concrete ore bin was built upon completion of the No. 10 tunnel and the dismantling of the aerial tram in 1924. The jaw crusher and the No. 6 K Gates crusher were also installed at this time. In 1926 the trommels were replaced by Hum-mer screens. These screens had 1¼-inch openings and the rolls were operated in closed circuit with them. This arrangement produced primary ball-mill feed of 1¼-inch maximum size; this size was the smallest that could be employed with the equipment in service. Mill tests, however, indicated that costs could be further decreased by feeding finer material to the primary grinding mills. As this change would mean a considerable expenditure for the installation of another fine-crushing machine, the plan was never completed.

In 1920 one 18-inch Minerals Separation flotation machine was added to supplement the 24-inch machine originally installed. Later, a 20 by 10 foot surge tank was added in order to maintain a more constant feed to the flotation circuit. A second 24-inch Minerals Separation machine was installed in 1922 to relieve the overloaded conditions of the 18-inch and original 24-inch machines. Scavenger pneumatic flotation cells were also added at this time for additional treatment of the tailings of the Minerals Separation machines.

The original equipment for cleaning rougher flotation middlings and concentrates of the primary machines comprised two pneumatic cleaners which handled middlings and one finishing pneumatic cleaner cell which handled the concentrates. The concentrates of the pneumatic cells handling middlings were recleaned in the finishing machine. The tailings from all pneumatic cleaner cells were returned to the heads of the primary roughers. The amount of sulphide minerals returned to the head of the primary roughers in the tailings products of the cleaner cells was considerably decreased in the latter part of 1921 by installing additional cells to operate in series with the pneumatic cleaners.

The tabulation which follows shows the average tonnage of ore milled per day, the average copper content of heads, the average grade of concentrates produced, the average extraction, and the average milling cost for each year of the period 1920 to 1929, inclusive.

Average tonnage milled per day, copper contents of heads, grade of concentrates, extraction and milling cost, 1920-1929

Year	Ore milled per day, tons	Assay of heads, per cent copper	Grade of concentrates, per cent copper	Extraction, per cent	Cost per ton of ore milled, dollars
1920	658.20	2.210	29.76	80.93	1.2407
1921	840.70	2.224	29.00	84.48	1.0332
1922	994.40	2.218	28.55	87.18	0.9310
1923	1,050.74	2.134	28.99	88.73	.8314
1924	962.76	2.054	27.89	89.40	.8486
1925	1,134.62	1.979	27.75	93.16	.7460
1926	1,164.70	1.785	24.32	88.66	.6778
1927	1,062.63	1.788	27.47	88.14	.6758
1928	985.90	1.710	27.79	91.58	.6780
1929	1,091.20	1.524	27.76	91.34	.6529

PRESENT METHOD OF CONCENTRATING

A plan showing the general arrangement of machines is presented in Figure 1, and a typical section of the concentrator is shown in Figure 2.

A general summary of method of treatment follows.

The ore is reduced to $1\frac{1}{4}$-inch size for ball-mill feed by primary and secondary jaw and gyratory breakers followed by rolls which operate in closed circuit with Hum-mer screens.

Grinding is done in two stages by ball mills.

The ground product contains 87 per cent of minus 100-mesh material and is treated in three Minerals Separation flotation machines which produce finished concentrates and three middlings products of different grades.

The middlings are retreated in pneumatic-type flotation machines. These cells produce finished concentrates and middlings which are returned to the head of the flotation circuit.

The concentrates, after dewatering, are loaded into gondola cars and shipped to the Garfield plant of the American Smelting and Refining Co.

The tailings pulp is conveyed in flumes a distance of $\frac{7}{8}$ mile down the canyon, where the coarser solids are removed. The slime pulp is conveyed an additional 4 miles in flumes and impounded.

Coarse Crushing

The flow sheet of crushing and grinding operations is presented in Figure 3.

The concentrator is provided with two coarse-ore storage bins, one for ore from each mine. Each storage bin is served by a separate coarse-crushing unit. The arrangement is flexible in that ore from one bin may be fed to the crushing unit which serves the other bin.

The size of openings in mine grizzlies is governed by the ability of the rock to flow freely through the chutes which follow the grizzlies. In the case of ore drawn directly from stopes into the mill ore train, occasional pieces are so large that they have to be plugged and either blasted or broken by key and feather before they can be fed into the crusher.

The bin for Superior mine ore is a 24 by 30 foot steel tank having a live capacity of about 500 tons. Ore is drawn through a 5 by 8 foot opening in the bottom of the bin by a 30-inch pan conveyor which feeds one of the coarse-crushing units.

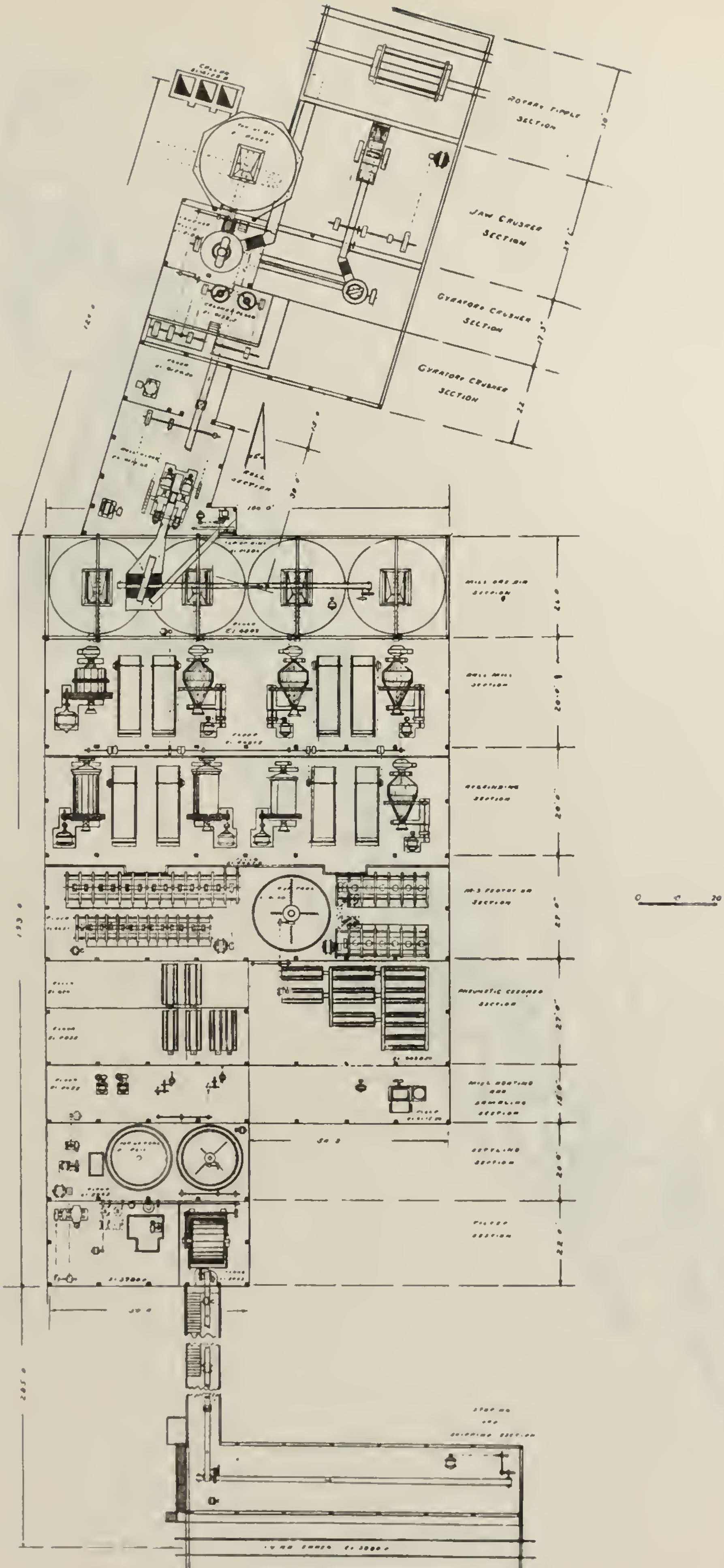

Figure 1.—General arrangement plan of Superior mill

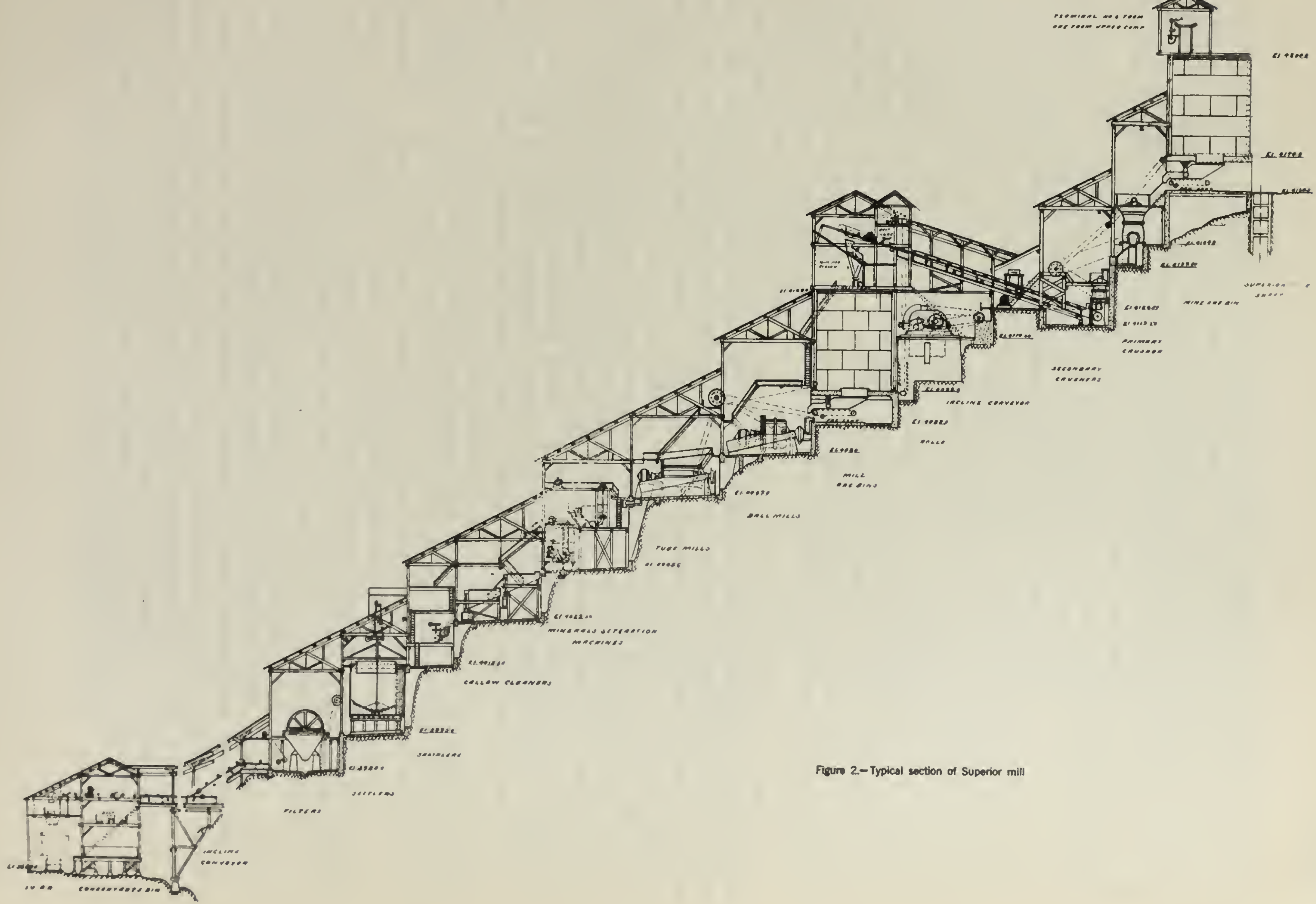

Figure 2.—Typical section of Superior mill

The storage bin for Engels ore is built of concrete, reinforced by 16-pound rails in the sides and 85-pound rails around the chute openings. The inside of the bin is lined with discarded manganese-steel ball-mill liners to prevent abrasion. The bin has a live capacity for 550 tons of ore. Two side openings are provided which allow ore to be fed to either of the two coarse crushing units. The feed to one of these units is drawn from the bin through an opening, 5 feet wide and 8 feet high, by a rotary feeder. The feed to the other unit is drawn through a corner opening, which is 4 feet wide and 8 feet high, by a pan conveyor. This latter opening is equipped with three 85-pound rails placed across the top and spaced 9 inches apart which makes the net opening 4 feet wide by 5 feet high.

The crushing unit for Superior ore comprises one No. 8 primary Traylor gyratory, which crushes to 3-inch size, followed by two secondary crushers, one a No. 5 Superior McCully and the other a No. 6 K Gates, which crush to approximately 2-inch size. The secondary crushers operate in parallel, each receiving as feed a portion of the primary-crusher product. The three crushers are driven from a line shaft which in turn is driven by a 150-hp. induction motor. When the three crushers are driven from the line shaft without loads the motor requires an input of 25 kilowatts and with full loads 64 kilowatts. The No. 8 primary crusher operates at a speed of 100 r.p.m., the No. 5 McCully at 125 r.p.m. and the No. 6 Gates at 140 r.p.m. The No. 5 McCully and the No. 6 K Gates crushers have capacities of 50 and 65 tons of ore per hour, respectively.

The crushing unit for Engels ore comprises one 24 by 36 inch Blake-type crusher set to crush to 3-inch size followed by one No. 6 K Gates machine which crushes to approximately 2-inch size. The primary-crusher product of this unit can be by-passed to the secondary crushers of the other unit by means of a 24-inch belt conveyor. The Blake crusher is driven at a speed of 222 r.p.m. from a line shaft by two 12-inch, 6-ply belts. At this speed it handles 65 tons of ore per hour. The secondary crusher is driven at a speed of 140 r.p.m. from the same line shaft and can handle 60 tons of primary-crusher product per hour, with the minus $1\frac{1}{4}$-inch material removed by a grizzly ahead of the machine. The line shaft for this unit is driven by a 150-hp. slip-ring motor. With both crushers operated at full load the motor takes from 90 to 125 kilowatts; without loads it requires 45 kilowatts. With the secondary crusher disconnected and the primary crusher operated at full load the motor takes 65 kilowatts and without load requires 36.5 kilowatts.

Grizzlies, 4 feet wide and with 3-inch spaces, are placed ahead of the primary crushers of both units. The undersize removed amounts to about 20 per cent of the total feed. The manganese-steel grizzly bars are 4 feet long and are set at an inclination of 34° to the horizontal. The secondary crushers are served by grizzlies with $1\frac{1}{2}$-inch spaces.

Both crushing units are operated at the same time when feed for both is available. As previously stated, the No. 8 primary machine can be fed from either ore bin and the ore feed is changed frequently during the shift from one to the other. The average tonnage of ore handled by the entire coarse-crushing plant amounts to 100 tons per hour.

The operating crew for coarse crushing comprises three men per shift. One man operates the feeders and watches the jaw and gyratory crushers in order to keep them running properly. Owing to the wet sticky nature of the ore from the Engels mine, a clean-up man is required to dispose of muck dropped from the return side of the conveyor belts. The third man breaks oversize pieces of ore at the concrete bin openings and also assists the clean-up man. At times the wet sticky ore requires additional labor to keep the plant functioning.

Table 1 gives a screen analysis of the coarse-crushing plant product.

Intermediate Crushing

The products of both coarse-crushing units join and, after passing over a Merrick weightometer, are fed to two Hum-mer screens which have 1¼-inch holes. These screens operate in closed circuit with a pair of 54 by 24 inch rolls. The rolls receive the oversize of the screens as feed and, after crushing, the product is returned to the screens by a bucket elevator and belt conveyor.

The undersize of the screens provides feed to the primary grinding mills and is distributed into four steel bins, one bin serving each grinding unit. Each bin is 24 by 30 feet, has a total capacity of 850 tons and a live capacity of 500 tons.

The rolls are spaced ¼ inch apart and are driven at a speed of 44 r.p.m. by belts from a line shaft which is in turn driven by a 100-hp. induction motor. Various roll speeds have been tried but when operated at a higher speed than present practice choking resulted which decreased capacity. New roll shells have an outside diameter of 55¾ inches and last from 4 to 6 months. The consumption of roll-shell steel amounts to 0.0716 pound per ton of ore crushed. Longitudinal corrugations form on the shells; these are reduced by grinding with carborundum bricks which are held in position by adjustable arms. When grinding the ridges it is necessary to use a small stream of water applied to the ridge to secure the maximum cutting effect.

One Hum-mer screen handles the product of the coarse-crushing plant and the other is placed in circuit with the rolls. The screens are inclined at an angle of 35° and each has an area of 20 square feet of screen surface. The vibratory magnets are supplied with current by a motor-generator set equipped with a 5-hp. motor. The power input of the motor is 1.5 kilowatts when operated without load and 2.6 kilowatts with load. Screens last about 30 days and magnet coils from 6 to 10 months.

One operator per shift takes care of the rolls, elevator, Hum-mer screens and distribution of the crushed ore to the bins. This operator is also responsible for the proper working of the entire crushing plant in the absence of the shift boss.

Table 1 gives screen analyses and tonnages of the feed, and intermediate and final products of the rolls section.

Table 1.--Screen analyses and tonnages of feed, intermediate and final products of the rolls section

Screen size	Coarse crushing plant product, new feed to Hum-mer screens, 100 tons per hour, per cent weight	Hum-mer screen oversize from coarse crushing-plant product, 60 tons per hour, per cent weight	Rolls discharge product, 124 tons per hour, per cent weight	Hum-mer screen oversize from rolls product, 64 tons per hour, per cent weight	Undersize products of both Hum-mer screens, 100 tons per hour, per cent weight
Plus 2 inch......	24.02	40.03	2.83	5.48	-
Plus 1.50 inch..	18.97	31.60	7.94	15.46	-
Plus 1.05 inch..	14.84	20.50	30.15	58.53	4.23
Plus 0.742 inch	9.86	7.21	23.10	18.85	22.16
Plus 0.525 inch	7.19	0.50	10.36	0.66	19.33
Plus 0.371 inch	5.45	[1]0.16	7.83	[1]1.12	14.39
Plus 3 mesh........	3.46	-	3.54	-	7.85
Plus 4 mesh........	2.33	-	3.01	-	6.07
Plus 6 mesh........	1.84	-	2.00	-	4.32
Plus 8 mesh........	1.59	-	1.59	-	3.56
Plus 10 mesh......	0.68	-	1.73	-	2.83
Plus 14 mesh......	0.73	-	1.00	-	1.97
Plus 20 mesh......	0.85	-	0.63	-	1.63
Plus 28 mesh......	0.54	-	0.46	-	1.11
Plus 35 mesh......	0.60	-	0.58	-	1.32
Plus 48 mesh......	0.69	-	0.49	-	1.30
Plus 65 mesh......	0.64	-	0.15	-	0.82
Plus 100 mesh....	1.13	-	0.34	-	1.25
Plus 150 mesh....	0.31	-	0.25	-	0.42
Plus 200 mesh....	1.12	-	0.47	-	1.20
Minus 200 mesh..	3.39	-	1.55	-	4.24

1 - Minus 0.525-inch size.

Grinding

There are four primary grinding mills, three of which are 8-foot by 36-inch Hardinge mills and one an 8 by 6 foot Marcy mill. Each mill, as previously mentioned, is served by a separate fine-ore bin. The feed to the grinding mills is drawn from the bottom of these bins and delivered to the mills by conveyor belts. Each primary mill is equipped with a combination scoop feeder and can be operated in either open or closed circuit with the Dorr classifier which receives the ground product. The mills are normally operated in open circuit, but when roll shells are being replaced they are operated in closed circuit.

Each primary mill is followed by a secondary grinding mill which receives as feed the sand product of the primary classifier. Each secondary mill is operated in closed circuit with a model-D Dorr classifier, the classifier sands being returned to the secondary mill. The combined overflow pulps of the primary and secondary classifiers comprise the feed to the flotation circuits.

The primary Hardinge mills operate at slightly different speeds but average 20.5 r.p.m. Each mill is driven by a 150-hp. slip-ring motor, through a 14-inch silent chain, and spur

gears. The average power input to each motor is 110 kilowatts. The ball load is maintained at about 12 tons by the addition of seven 5-inch balls during each shift. The consumption of balls amounts to 1.032 pounds per ton of new feed. Annealed steel liners are held in place by wedge bars of the same material, and the liner consumption is 0.231 pound per ton of ore milled. A chemical analysis of liner material follows.

	Per cent
Silicon	0.24
Chromium	0.15
Phosphorous	0.028
Manganese	0.62
Combined carbon	0.92
Sulphur	0.051

The tabulation which follows gives the average life of each row of liners and wedge bars over a number of years.

Average life of liners and wedge bars

Location of row	Liners		Wedge bars	
	Row	Life, days	Row	Life, days
Feed end of mill	A	170	-	-
Feed end of mill	B	242	B	158
Cylinder	C	204	C	170
Discharge side of cylinder	D	192	D	178
Discharge side of cylinder	E	408	E	289
Discharge end of mill	F	640	-	-

The mills are stopped and inspected every two weeks of running time to replace thin or worn-out liners. This procedure is strictly followed to prevent undue wear of the mill shell. In spite of this precaution, it has been found that the shell becomes worn by pulp flowing between the ends of the liner rows and also between the "A" row of liners and the shell which necessitates replacement of sections of the shell about every five years of running time. Some experimental work is being done with a rubber lining between the shell and the regular liners, but definite information as to the value of this procedure has not yet been obtained. The combination scoop feeders last about three years and the spur pinions about four years.

The pulp in the Hardinge primary mills is maintained at 80 per cent of solids and each mill has an average capacity of 382 tons of ore per day when operated in open circuit.

The No. 86 primary Marcy mill is driven at a speed of 21 r.p.m. by a 225-hp. slip-ring motor which is coupled to the pinion shaft of the mill by double disks and 10 rubber bushings. The gear and pinion are of the Fawcus type. The power input to the mill is 200 kilowatts. The Marcy mill receives about one-half of the sand product of the primary Dorr classifier in addition to the new ore feed; the balance of the sand from the primary classifier provides feed to the secondary grinding mill of this unit. When operated in this manner the primary mill handles 588 tons of new feed and 200 tons of return sands per day. The mill is operated with a ball load of 12.5 tons and this load is maintained by the addition of twelve 5-inch balls during each shift. The consumption of balls amounts to 1.176 pounds per ton of new feed. Forged-steel and cast-steel balls have been compared over long periods with the

result that the cast-steel balls showed a lower cost per ton of ore milled. Manganese-steel liners of the block type with corrugated or waved surfaces are used. The liner consumption averages 0.1814 pound per ton of new feed. The pulp in the mill is maintained at about 83 per cent of solids. The feeder scoop lasts about four years and the pinion about two and a half years.

The secondary grinding mills comprise two 6 by 12 foot Power and Machinery Co. ball-tube mills, one Power and Machinery Co. 6 by 10 foot mill which, as previously stated, was converted from a 7 by 10 foot size machine, and one 8-foot by 36-inch Hardinge mill.

Each 6 by 12 foot machine is driven at a speed of 24 r.p.m. by a 150-hp. slip-ring motor through a Falk-type pinion and gear. The pinion shaft is connected to the motor by the overlapping disk and belt-lacing type of coupling. The power input of each motor is 120 kilowatts. The average ball load amounts to 14 tons and is maintained by the addition of 130 pounds of 2½-inch chilled cast balls during each shift. Liners are of chilled iron and are cast with two small lengthwise ridges on each liner. Liner consumption is 0.1052 pound per ton of new feed handled. Pulp consistency in the mills is maintained at 76 per cent of solids. As previously stated, these mills operate in closed circuit with Dorr classifiers. Each mill is equipped with a 2-point feed scoop and carries a heavy circulating load. Pinions last about four years, scoops about five years, and cast-iron screens from six to eight months of operating time.

The 6 by 10 foot mill is driven at a speed of 23 r.p.m. by an old-style 85-hp. slip-ring motor which takes a power input of 96 kilowatts. The ball load is maintained at about 7½ tons by the addition of 80 pounds of balls per shift.

The secondary Hardinge mill is driven at a speed of 17 r.p.m. by a 150-hp. slip-ring motor which requires 75-kilowatts input. The ball load is maintained at about 12 tons by the addition of 100 pounds of balls per shift.

All classifiers used in both primary and secondary grinding circuits are model-DSD Dorr machines, except one model-C machine which is operated with one of the primary Hardinge mills The model-DSD classifiers are 23½ feet long by 6 feet wide and the model-C is 20½ feet long and 4½ feet wide. They operate at a slope of 2⅝ inches per foot and with rake speeds of 22 strokes per minute. All classifiers and primary grinding-mill feeders are driven from a line shaft which in turn is driven by a 35-hp. motor. The power input to the motor when the line shaft is running and all belts are on idler pulleys is 4 kilowatts. This input increases about 0.8 kilowatts for each classifier operated. It is not believed advisable to reduce the size of the driving motor, since it appears to be well loaded in starting up machines after brief power interruptions. A power interruption of 9 to 10 minutes is usually sufficient to clog a classifier so that it must be drained in order to start. Classifier pulps of the primary Hardinge units are maintained at 50 per cent of solids, 43 per cent in the primary Marcy unit and 45 per cent in the secondary grinding units.

Two men per shift are required for the operation of the primary and secondary grinding circuits. The feed to a primary mill is regulated largely by the sound of the balls in the mill and mills are operated at their maximum capacity as much as possible.

Some interesting facts regarding the operation of a Marcy mill at this plant have been worked out. When the primary Marcy mill was operated in open circuit and all classifier sands were sent to the secondary grinding circuit, the capacity of the Marcy mill was 670 tons per day, but this method of operation required two 6 by 12 foot secondary mills to finish the grinding of the Marcy sands. When the Marcy mill operated in partial closed circuit and handled 588 tons of new feed and in addition 200 tons of return sand, one 6 by 12 foot secondary mill was able to complete the grinding.

The operation of the primary Hardinge mills in partly closed circuit was not tried, principally because of the fact that the ball-tube secondary mills, in series with them, were working at their maximum efficiencies with the primary mills operated in open circuit.

Table 2 presents typical tonnages and screen analyses of the feed, intermediate, and final products of the primary Marcy mill circuit. Table 3 gives typical tonnages and screen analyses of feed, intermediate, and final products of the secondary-mill grinding circuit which is in series with the primary Marcy unit.

Table 4 presents similar data for the primary Hardinge mill units, and Table 5 for the secondary grinding units which operate in series with the primary Hardinge units.

Table 2.—Typical tonnages and screen analyses of the primary Marcy mill grinding-circuit products

Screen size	Marcy mill new ore feed, 24.5 tons per hour	Classifier sands returned to the Marcy mill, 8.33 tons per hour	Marcy mill discharge, 32.83 tons per hour	Classifier sands sent to secondary grinding as tube mill feed. 12.05 tons per hour	Classifier overflow, 12.45 tons of solids per hour
	Weight, per cent	Weight, per cent	Weight, per cent	Weight, per cent	Weight, per cent
Plus 1.05 inch..	1.83	-	-	-	-
Plus 0.741 inch	19.45	-	-	-	-
Plus 0.525 inch	18.30	-	-	-	-
Plus 0.371 inch	16.59	-	-	-	-
Plus 3 mesh........	9.38	-	-	-	-
Plus 4 mesh	5.99	-	-	-	-
Plus 4 mesh	5.99	-	-	-	-
Plus 6 mesh... ..	4.25	-	-	-	-
Plus 8 mesh	3.49	3.66	2.36	3.71	-
Plus 10 mesh......	4.03	5.15	2.97	4.34	-
Plus 14 mesh......	1.88	3.00	4.36	6.16	-
Plus 20 mesh......	1.53	15.10	7.95	11.01	-
Plus 28 mesh......	1.02	20.63	9.00	10.22	-
Plus 35 mesh......	1.14	8.09	6.95	13.34	0.02
Plus 48 mesh......	1.03	8.58	6.91	12.15	0.73
Plus 65 mesh......	0.92	7.33	6.06	9.18	2.09
Plus 100 mesh....	1.34	8.49	9.46	10.16	9.48
Plus 150 mesh....	0 94	2.29	3.77	2.78	5.72
200 mesh....	2.09	4.92	10.85	6.26	19.26
Minus 200 mesh.	4.80	7.56	29.36	10.69	62.70

Table 3.--Typical tonnages and screen analyses of products from the secondary grinding mill of the circuit which operates in series with the Marcy unit

Screen size	Feed to tube mill, 12.05 tons per hour	Tube mill discharge, 50.00 tons per hour	Classifier overflow, 12.05 tons of solids per hour	Classifier return sands to tube mill, 37.95 tons per hour
	Weight, per cent	Weight, per cent	Weight, per cent	Weight, per cent
Plus 8 mesh	3.71	0.17	-	0.22
Plus 10 mesh	4.34	0.22	-	0.29
Plus 14 mesh	6.16	0.49	-	0.64
Plus 20 mesh	11.01	0.81	-	1.07
Plus 28 mesh	10.22	2.06	-	2.75
Plus 35 mesh	13.34	5.23	-	6.90
Plus 48 mesh	12.15	10.49	0.06	13.81
Plus 65 mesh	9.18	15.00	1.94	19.15
Plus 100 mesh	10.16	18.94	7.33	22.61
Plus 150 mesh	2.78	7.82	5.42	8.58
Plus 200 mesh	6.26	16.05	19.90	14.84
Minus 200 mesh	10.69	22.72	65.35	9.17

Table 4.--Typical tonnages and screen analyses of the primary Hardinge mill grinding-circuit products

Screen size	Hardinge mill new ore feed 15.9 tons per hour	Hardinge mill discharge 15.9 tons per hour	Classifier overflow, 6.85 tons of solids per hour	Classifier sands sent to secondary grinding circuit as tube-mill feed, 9.05 tons per hour
	Weight, per cent	Weight, per cent	Weight, per cent	Weight, per cent
Plus 1.05 inch	1.66	-	-	-
Plus 0.741 inch	15.91	-	-	-
Plus 0.525 inch	17.90	-	-	-
Plus 0.371	15.25	-	-	-
Plus 3 mesh	11.94	-	-	-
Plus 4 mesh	5.83	-	-	-
Plus 6 mesh	4.09	-	-	-
Plus 8 mesh	3.34	1.44	-	2.63
Plus 10 mesh	3.32	1.94	-	3.41
Plus 14 mesh	2.06	2.44	-	4.28
Plus 20 mesh	1.71	4.35	-	7.65
Plus 28 mesh	1.20	9.41	-	16.39
Plus 35 mesh	2.98	8.52	-	15.00
Plus 48 mesh	2.65	7.36	0.90	12.25
Plus 65 mesh	1.25	7.65	4.79	9.82
Plus 100 mesh	1.61	9.16	12.19	6.88
Plus 150 mesh	0.66	4.90	7.36	3.05
Plus 200 mesh	2.65	11.74	19.97	5.61
Minus 200 mesh	3.99	31.09	54.79	13.03

Table 5.--Typical tonnages and screen analyses of products from the secondary grinding circuits which operate in series with the Hardinge primary mills

Screen size	Feed to tube mill, 9.05 tons per hour	Tube mill discharge, 40.00 tons per hour	Classifier overflow, 9.05 tons of solids per hour	Classifier return sands to tube mill, 30.95 tons per hour
	Weight, per cent	Weight, per cent	Weight, per cent	Weight, per cent
Plus 3 mesh	2.63	-	-	-
Plus 10 mesh	3.41	-	-	-
Plus 14 mesh	4.28	-	-	-
Plus 20 mesh	7.65	-	-	-
Plus 28 mesh	16.39	0.56	-	0.71
Plus 35 mesh	15.00	2.43	-	3.14
Plus 48 mesh	12.25	6.16	0.16	7.91
Plus 65 mesh	9.82	10.26	1.81	12.71
Plus 100 mesh	6.88	19.47	7.08	23.04
Plus 150 mesh	3.05	7.52	5.77	8.03
Plus 200 mesh	5.61	22.60	19.82	23.38
Minus 200 mesh	13.03	31.00	65.32	21.08

Flotation

The flow sheet of flotation and dewatering operations is presented in Figure 4.

The overflow pulps from the classifiers of the grinding units flow to a surge tank, which feeds three 12-cell Minerals Separation machines. These machines operate in parallel and produce concentrates froths which are divided into four grades for subsequent treatment. The first cell of each machine produces finished concentrates which go directly to the concentrates thickener. The froth products of cells 2 to 4, inclusive, are cleaned in a Callow-type pneumatic machine referred to as the "finisher" machine, which produces finished concentrates and middlings that join the froth products of Mineral Separation cells 5 to 7, inclusive, for two stages of cleaning in Callow cells. The primary Callow machine of this cleaning unit produces concentrates which join the froth products of M. S. cells 2 to 4 for additional cleaning in the "finisher" machine, and middlings which join the froth products of M. S. cells 8 to 12, inclusive, for treatment in the secondary Callow cleaner of the unit. The froth product of this secondary cleaner is returned to the head of the primary Callow cleaner and the middlings are returned to the surge tank at the head of the Minerals Separation circuit. The tailings of the Minerals Separation machines are given final flotation treatment in six Callow-type cells operated in series as scavenger units. These cells produce froth products, which are returned to the surge tank at the head of the Minerals Separation machines, and tailings which are waste products.

The surge tank at the head of the flotation circuit is 20 feet in diameter and 10 feet high. A vertical cylinder, 2 feet in diameter, is placed in the center of this tank for receiving feed pulp. The bottom of this cylinder is placed 2 feet above the bottom of the tank. Two 6-inch centrifugal pumps, one of which is used as a spare, circulate the pulp from the bottom of the surge tank to a launder which returns the pulp to the top of the surge tank. This launder is equipped with a splitter which deflects a portion of the circulating

pulp to the flotation machines. The splitter is operated by a float within the surge tank. When the input of pulp to the surge tank increases, the float rises and opens the splitter, thus sending more pulp to the flotation machines and returning less to the surge tank.

The pulp level within the surge tank has a maximum range of 3½ feet. This range gives the flotation operator ample time to adjust the flotation machines when grinding mills are started or stopped and thus prevents sudden changes in the rate of pulp flow through the flotation machines and, in addition, maintains a more uniformly conditioned pulp. Rakes were formerly operated in the bottom of the surge tank but were found to be unnecessary. The circulating pump of the surge tank elevates the pulp a distance of 20 feet to the launder and is driven at a speed of 860 r.p.m. by a direct-connected 25-hp. motor which takes a power input of 11.5 kilowatts.

The Minerals Separation machines, which comprise two 24-inch machines and one 18-inch machine, are of the spitz type. The 24-inch machines are driven by belts from 60-hp. and 75-hp. motors which take power inputs of 50 and 56 kilowatts, respectively. These machines have impeller speeds of 260 r.p.m., and each will handle 550 tons of ore per day when the pulp contains 26 per cent of solids. The 18-inch machine is chain-driven by a 50-hp. motor which takes a power input of 37 kilowatts. The impellers operate at a speed of 315 r.p.m. This machine is of the deep-cell type and will treat 300 tons of ore per day when the pulp contains 26 per cent of solids.

All pneumatic cells are of the Callow type. Six of these cells are operated as cleaners six as scavengers, and three additional scavenger cells are held in reserve as spare units. The scavenger cells are operated in three stages. The first stage contains three cells, the second stage two cells, and the final stage one cell. Each pneumatic cell is 8 feet long and 2 feet wide. The bottom of the cell slopes from a depth of 16 inches at the head end to 40 inches at the foot. Canvas blankets are made of 3-ply material and are stretched over 2 by 1 foot cast-iron pans. The air inlets are located at the bottom of the cell. The air is furnished by a No. 8 Sturtevant positive blower, which operates at a speed of 170 r.p.m. Each cell consumes 135 cubic feet of free air per minute at 4½ pounds pressure, which is equivalent to 1.90 hp.

Two 4-inch centrifugal pumps, belt-driven by 20-hp. motors, handle the pulps in the cleaner flotation circuit. Either pump can be used to lift the return pulps to the surge tank, a vertical distance of 24 feet. When one pump is being repaired, the finisher cell middlings and secondary cleaner cell froths are sent, with the cleaner cell middlings, to the surge tank. The pumps operate at a speed of 1,150 r.p.m. One set of liners usually lasts as long as two runners, which is about 180 days. With one pump elevating pulp to the surge tank and one elevating pulp to the rougher, the power input to the motors is 8¾ and 6.6 kilowatts, respectively.

A fuel-oil collector and a pine-oil frother were used as flotation reagents until the introduction of xanthate. Since the addition of xanthate into the flotation circuit, better results have been obtained. The addition of two kinds of pine oil, added separately to the pulp, has recently been found to give better results than the use of either alone. One of the pine oils makes a brittle froth and the other a fairly tough froth, so that the froth is easily controlled by varying the amounts of oils fed. The use of two frothing oils also made possible the substitution of lime for trisodium phosphate.

The table which follows gives the average amounts and kinds of flotation reagents used with lime and trisodium phosphate, respectively, during 1929.

Average amounts and kinds of flotation reagents used with lime and trisodium phosphate during 1929

	Pounds per ton of ore milled	
	Using lime	Using trisodium phosphate
Lime	0.1806	-
Trisodium phosphate	-	0.0600
Xanthate	0.0787	0.0753
Fuel oil	0.1045	0.2402
M. S. No. 14 pine oil	0.1571	0.1788
Risor pine oil	0.0485	0.0604
Total	0.5694	0.6147

Lime is crushed to minus ¼-inch size and is fed in the dry condition to the ball mills from a hopper by a rotary feeder. The lime feeder is driven from the conveyor which feeds the grinding mills. The amount of lime fed is enough to produce an alkaline pulp with phenolphthalein indicator. The pH value of the pulp water amounts to about 8.2.

When trisodium phosphate was used a batch of the latter was charged into a mixing tank daily and the resulting solution was introduced at the ball mills. The amount fed was determined by trial, the lower limit being indicated by erratic flotation results.

Xanthate solution, which contains 5 per cent of xanthate by weight, and all oils are stored in 100-gallon capacity tanks, and pipe lines connect these tanks to the various points of feeding.

Fuel oil is fed through a petcock to a small plunger pump which in turn feeds this reagent to the primary ball mills. Xanthate is added to the pulp at the head cell of each Minerals Separation machine, and a very small additional amount is added to the pneumatic finisher cell. The tough frothing pine oil is fed to the head cell of each Minerals Separation machine, and small amounts of the brittle frothing pine oil are added at the same points and to the seventh cells as needed.

The reagent tanks are equipped with floats and gages which give the operator a quick check on the consumption of reagents. The additions of reagents are governed entirely by the appearance of the froth.

Pulp density in the Minerals Separation machines is maintained between 26 and 28 per cent of solids. The dilution of pulp encountered between the classifiers and the M. S. machines is due to the return middlings pulp. Water must be added to break down the froth and also to dilute the middlings in order to obtain a satisfactory grade of concentrates in the cleaner cells. The middlings pulp returned to the surge tank averages 7.5 per cent of solids. Pulp densities are determined twice per shift by the flotation operator. The pulp is placed in a conical container and weighed on a vender's-type spring scale equipped with a graduated scale which gives direct density readings.

The flotation concentrates average 32 to 34 per cent of insoluble. The reduction of insoluble content without appreciable loss of copper can be accomplished only by additional grinding and floating.

The Minerals Separation machines produce 64.94 per cent of the total concentrates, which contains 66.70 per cent of the total copper recovered. The cleaner cells are operated from the viewpoint of removing the maximum amount of sulphides from the circuit, consistent with suitable grade, before returning the pulps to the Minerals Separation machines. The loss of copper in the final tailings varies directly with the copper content of the return middlings.

From data obtained in this plant it has been found that the pneumatic cells give much better results than mechanical cells when operated as cleaners, owing to greater flexibility and ease of control of the former.

Table 6 gives typical amounts and copper contents of the concentrator feed, the intermediate, and final products. Table 7 presents sizing assay tests of typical concentrator heads, concentrates, and tailings and also gives indicated recoveries of copper for each screen size based on heads and tailings assays.

Table 6.--Typical amounts and copper contents of concentrator products

Product	Weight, tons per 24 hours	Copper, per cent
New ore fed to Minerals Separation machines	1,091.00	1.54
Finished concentrates from No. 1 cells of M. S. machines	33.57	29.18
Concentrates from M. S. cells 2 to 4, inclusive, sent to finisher cell	39.75	13.42
Concentrates from M. S. cells 5 to 7, inclusive, sent to primary cleaner	43.43	3.13
Concentrates from M. S. cells 8 to 12, inclusive, sent to secondary cleaner	34.56	1.92
Tailings from M. S. machines, sent to scavenger cells	1,055.98	0.16
Finisher-cell concentrates, finished concentrates	18.12	27.00
Finisher-cell tailings, sent to primary cleaner	53.16	12.18
Primary-cleaner concentrates, sent to finisher cell	31.53	19.98
Primary-cleaner tailings, sent to secondary cleaner	76.40	3.37
Secondary-cleaner concentrates, sent to primary cleaner	11.34	11.22
Secondary-cleaner tailings, sent to head of M. S. machines	99.62	1.96
Scavenger-cell concentrates, sent to head of M. S. machines	16.67	1.05
Scavenger-cell tailings, final waste product	1,039.31	0.15

Table 7.--Typical screen analyses of concentrator heads, concentrates, and tailings

Screen size, mesh	Heads		Concentrates		Tailings		Indicated recovery based on heads and tailings, per cent
	Weight, per cent	Assay, per cent of copper	Weight, per cent	Assay, per cent of copper	Weight, per cent	Assay, per cent of copper	
Plus 65	2.80	0.47	0.35	7.86	2.57	0.28	45.31
Minus 65 plus 100	10.79	0.53	4.52	10.43	11.03	0.21	59.50
Minus 100 plus 150	4.91	0.82	3.55	11.60	5.10	0.15	81.00
Minus 150 plus 200	19.08	1.20	17.50	18.00	20.54	0.12	89.23
Minus 200	62.42	1.94	74.08	32.90	60.76	0.15	94.73
Composite	100.00	1.55	100.00	28.41	100.00	0.15	90.08

DEWATERING OF CONCENTRATES

The finished concentrates pulps from the flotation machines, which contain 37.5 per cent of solids, flow into a 16 by 16 foot Dorr thickener. This thickener has a froth retaining

wall built for a distance of 2 feet above the top of the tank. The overflow of the thickener goes to a 16-foot diameter settling tank which is equipped with a conical bottom; the overflow of this tank is returned to the surge tank at the head of the flotation machines. The thickener underflow, which contains 75 per cent of solids, is discharged periodically into the filter. The thickener rakes are driven at a speed of 1 revolution in 2¼ minutes by a 7½-hp. motor which requires a power input of 2.35 kilowatts. A 12-foot derrick is built over the thickener drive wheel, and the rakes can be raised as much as 7 feet by a set of 3-ton chain blocks which are suspended from this derrick.

The 11½ by 8 foot Oliver continuous filter is operated at a speed of one revolution in 7½ minutes. The feed pulp is heated by live steam to a temperature of 80° C., as it has been found that without this heating of the pulp, the filter cloth becomes so badly choked that it is useless after a run of one week's duration. This choking takes place even after steaming and scrubbing the cloth twice each day between shifts.

An average vacuum of 21 inches of mercury is maintained by a Doak 14 by 8 inch reciprocating pump which operates at 250 strokes per minute. A 15-hp. motor drives the filter and the vacuum pump from a line shaft and requires a power input of 5.72 kilowatts when operated under load. A "blow" of air is given to each filter section as it reaches the scraper and is maintained for two sections below the scraper. The filter cloth is made of loose twilled threads and dewaters 12,000 tons of concentrates or 416.64 tons of concentrates per square foot before being discarded. The life of the cloth is dependent on the life of the winding wire, as the scraper blade cuts the cloth when the wire wears thin.

The filter is operated on two shifts per day and has capacity to dewater from 3 to 10 tons of solids per hour, according to the amount of slimes contained in the pulp. This capacity amounts to from 0.00108 to 0.00347 ton of solids per square foot of filter area. The moisture content of the filter cake averages between 10 and 11 per cent.

DEWATERING AND DISPOSAL OF TAILINGS

The tailings plup flows in a flume to one of two dewatering plants. These dewatering plants, termed "upper" and "lower," are used alternately, the former being used principally during the summer months.

The flume which conveys the tailings to these units is 4,600 feet long and is set at grade of ¼-inch to the foot. The bottom of the flume is 5¼ inches wide and the sides have a slope of 4 inches per foot vertically.

The upper dewatering unit is equipped with two model-C Dorr classifiers which remove the coarse material. The classifier sands are conveyed by launder to a bucket elevator which in turn discharges the material to a V-bottom flume. The flume, which is set with a grade of 3 inches per foot, conveys this pulp to the coarse-tailings reservoir.

The lower dewatering unit is similar to the upper except that three 6 by 20½ foot Akins classifiers are used for separating the coarse material from the pulp.

The classifier overflow pulps of either dewatering unit go to an 80-foot Dorr thickener. This thickener produces a clear overflow which is discharged into Lights Creek and an underflow which is conveyed by flume for a distance of 4 miles to the slime-storage dam.

The Dorr classifiers of the "upper" unit operate with a rake speed of 14 strokes per minute and with a bottom slope of 1½ inches per foot. The consistency of the sand pulp is controlled by adding or removing cross pieces which are fitted to the classifier sand discharge.

The bucket elevator which delivers the sand product to the flume is 38 feet high and is equipped with a 12-inch 7-ply rubber covered belt which is driven at a speed of 345½ feet per minute by a 10-hp. motor. This belt operates over head and tail pulleys which are 30

inches in diameter and is equipped with 10 by 6 by 6 inch buckets spaced 7 inches apart. Two $\frac{1}{4}$-inch holes are drilled in the bottom of each bucket to prevent the sand from settling and sticking.

The Dorr thickener shaft is driven at a speed of 1 revolution in 13 minutes by a $7\frac{1}{2}$-hp. motor. Lime is added to the thickener feed pulp as it leaves the drag classifiers in sufficient amount to produce a clear overflow at the thickener. The lime feeder consists of a 2-cubic foot capacity hopper equipped with a 4 by 6 inch reciprocating plunger which operates across the bottom. The plunger has a 6-inch opening and discharges 140 cubic inches of freshly slaked lime every 5 minutes. The feeder discharges the slaked lime into a 4 by 4 foot mixing tank. A stream of water flows continuously into this tank, and the charge is mixed by a revolving iron weight supported at the end of a shaft. The amount of lime fed per shift is governed by the frequency with which the feed hopper is filled. The average amount fed is 0.614 barrel per 100 tons of tailings.

The reservoir for coarse tailings has an area of 14 acres. The retaining dam is built of sand by a 30-hp. caterpillar and a $1\frac{1}{4}$-yard self-dumping scraper. This dam is 6 feet wide at the top and is usually maintained from 2 to 5 feet higher than the level of the settling pond. The top of the tailings pile slopes about 2 inches per foot in a direction away from the dewatering plants.

Cross dams are built every 300 feet and these serve as approaches by which the tractor and scraper travel on and off the outside dam. The dam has an earth core about 8 feet high and the sand dam is built to a height of 60 feet above this core. A toe of mine rock, 20 feet high, supplemented by log cribbing protects it from the high water of Lights Creek. The earth core is impervious to water, and this caused trouble during the winter months by making a plastic mass of the sand, which caused portions of the dam to slough off. This difficulty was avoided by driving a number of 2-inch pipes, having screened well points at the ends, into the dam at points low enough to maintain the water level below the earth core. No sloughing has been experienced since the introduction of this method of draining. It has been also found necessary to build up the dam for winter storage before freezing weather sets in. Constant attendance is required only during the spring thaw when the water from melting snow and ice washes the sand rapidly down to the dam and causes choking in the overflow boxes, if not watched. Brush is placed along the top of the dam on the prevailing wind side to prevent wind cutting. The height of water in the pond is maintained as low as possible consistent with obtaining a clear overflow. During the summer months there is very little overflow water owing to evaporation, but the condition is otherwise during the winter months, and lime is required for settling. One man with a tractor can build about 300 cubic yards of dam per shift.

The storage dam for slime controls an area of 47 acres. This area is divided into four sections by cross dams. When a section has filled to such a height that it becomes necessary to raise the dam, the slime pulp flow is changed to another pond. It requires from four to six months after this for the slime to dry sufficiently to allow a scraper to be used over it.

SAMPLING

A sample of the heads is taken by a mechanical sampler from the classifier overflow pulp just before it reaches the surge tank at the head of the flotation machines. The cutter crosses the pulp stream every 14 minutes, and when the concentrator is treating 1,000 tons of ore per day, it cuts a sample which contains about 4 pounds of solids per shift.

The tailings sample is obtained as the pulp enters the main flume. The cutter crosses the pulp stream every 9 minutes and obtains a sample which contains about $3\frac{1}{2}$ pounds of solids per shift.

A sample of concentrates is obtained just as the pulp enters the thickener. The pulp stream is cut every 7 minutes and the weight of solids in this sample per shift amounts to about 4 pounds. A grab sample of concentrates is also taken periodically from the filter cake by the operator for checking purposes. A third concentrates sample is obtained from the gondola. After a gondola is loaded the top is leveled and borings obtained at 12 points with a 1½-inch auger. The samples are taken at regular intervals in two rows along the length of the car. This sample is used for determinations of moisture, copper, silver, gold, and insoluble contents.

The cutters of all mechanical samplers are actuated by weights. This method of operation is used to prevent the spoiling of samples by power interruptions which cause the cutter to stick in the stream.

Samples are dried each shift on steam hot plates and sent to the assay office each morning. Assay returns are reported to the mill usually before the day shift is over. A sizing test is made on the head sample of each shift to keep a check on classifier operation. The concentrates sample is assayed for insoluble content in addition to copper to aid the operator in judging the grade being produced.

WATER SUPPLY

As previously stated, sufficient water for concentrator operations is usually available from surface sources. This water flows by gravity to a 50,000-gallon capacity storage tank. When a shortage of surface water develops, the deficiency is made up by using mine water from the Engels mine. When mine water is used, trisodium phosphate must be fed to the ball mills to insure uniform results in flotation, as the mine water, although not acid, apparently contains impurities which affect flotation operations adversely. The consumption of new water amounts to 680 gallons per ton of ore milled.

METALLURGICAL DATA

Table 8 presents metallurgical data for the year 1929.

Table 8.--Metallurgical data for the year 1929

Total ore milled	tons	395,042
Operating time per day	hours	24
Average ore milled per 24 hours	tons	1,091.30
Total concentrates produced	tons	19,813.08
Average concentrates produced per 24 hours	tons	54.73
Recovery of total copper	per cent	91.34
Ratio of concentration	tons into 1	19.94
Consumption of 5-inch balls per ton of ore milled	pounds	1.104
Consumption of 2½ inch balls per ton of ore milled	pounds	0.72
Total ball consumption per ton of ore milled	pounds	1.824
Liner consumption per ton of ore milled	pounds	0.32
Insoluble content of concentrates	per cent	34.52
Material plus 100-mesh size	per cent	12.64
Average lime used per ton of ore to settle tailings	barrels	0.00614

CONVEYING

The No. 8 gyratory crusher is fed, as previously stated, from either of two ore bins, each of which is served by a pan conveyor. The shock of dumping rock into an empty bin is taken up by swing hammers. One conveyor, which is 30 inches wide, 12.5 feet long, and which operates at a speed of 4.4 feet per minute, draws mine ore from the bottom of the steel bin. The other, which is 36 inches wide, 36.5 feet long, and which operates at a speed of 7.5 feet per minute, draws mine ore from the corner of the concrete bin. The pans of each conveyor have 1½-inch wood fillers which are covered by mild-steel plates 1½ inches thick. Each conveyor operates at a capacity of 90 tons of ore per hour.

The Blake-type crusher is fed from the concrete bin by a 30 by 60-inch rotary feeder which operates at a speed of 0.455 r.p.m. The storage-bin discharge opening is equipped with five 4 by 6 inch fingers which permit barring without gushing of ore when the opening becomes choked. These fingers are lined with manganese-steel plates where contact is made with the ore and are otherwise similar to the fingers used in underground chutes. The crushed product of the Blake machine drops onto a 24-inch rubber-covered conveyor belt which travels at a speed of 302 feet per minute. This belt is 75 feet long and elevates the ore a distance of 5 feet. With a 3/16-inch rubber cover it has a life of about 720,000 tons of ore.

The pan conveyors, rotary feeder, and the conveyor which handles the product of the Blake crusher are driven from line shafts which drive the respective crushers.

The products from the No. 5 McCully and the No. 6 K Gates crushers are conveyed to the Hum-mer screens by a 24-inch rubber-covered belt which travels at a speed of 285 feet per minute. This belt is 180 feet long, passes over a Merrick weightometer, and elevates the ore a distance of 28 feet. It is driven by a 10-hp. motor which requires a power input of 4.4 kilowatts without load and 6.8 kilowatts at full load. With a 3/16-inch rubber cover the life of this belt is 1,400,000 tons of ore.

The oversize from the Hum-mer screens is fed to the rolls by a 28-inch belt which is 6 feet long and which travels at a speed of 14.5 feet per minute. It is equipped with side boards, 2½ feet high, to hold the material transported. It has a life of about 900,000 tons or ore.

The roll product is returned to the Hum-mer screens by a bucket elevator and conveyor. The bucket elevator is 55 feet high and slopes 80° from the horizontal. It is equipped with a rubber-covered 10-ply 28-inch belt and buckets 20 by 8 by 8½ inch spaced 14 inches apart lip to lip. The belt is driven at a speed of 306 feet per minute by a 20-hp. motor which requires a power input of 8.5 kilowatts without load and 17 kilowatts with full load. This elevator will handle 150 tons of ore per hour, and its belt has a life of 1,700,000 tons of ore.

The 22-inch conveyor which receives the roll product from the elevator and delivers it to the Hum-mer screens is 30 feet long and is driven at a speed of 245 feet per minute by a 5-hp. motor which requires a power input of 1.5 kilowatts without load and 4.6 kilowatts with full load. It has a life of about 1,700,000 tons of ore.

The undersize product of the Hum-mer screens is transported to the ball-mill feed bins by a 16-inch rubber-covered belt. This belt is driven at a speed of 310 feet per minute by a 10-hp. motor which consumes 3.5 kilowatts without load and 5.5 kilowatts with full load. The average life of this belt is 800,000 tons of ore. It is equipped with a movable distributor for unloading and delivering the ore to the desired bin.

The ball mills are fed by 24-inch belts equipped with covers ⅛-inch thick. The feeder belts which serve the Hardinge mills operate at a speed of 13 feet per minute and have a life of about 400,000 tons of ore. The Marcy-mill feeder belt travels at a speed of 22 feet per minute and also has a life of about 400,000 tons of ore. Two of the ball mills were originally fed by 18-inch pan conveyors. The upkeep of these pan conveyors was so high that rubber belts were substituted when the pans wore out.

Concentrates are transported down to the railroad cars by three belts arranged in series The first belt is 14 inches wide by 155 feet long, has a drop of 80 feet, and travels at a speed of 150 feet per minute. Due to the steep slope of the belt, it is necessary to enclose it with boards to keep the concentrates from rolling off. Sacks are suspended every 6 feet from the underside of the top cover to further check this rolling action. The second belt is 12 inches wide, 170 feet long, and operates at a speed of 66 feet per minute. This belt is operated at sufficiently low speed to give it the proper load per square foot for the operation of the weightometer over which it passes. The concentrates pass finally to the third belt which runs at right angles to the second belt and passes over the railroad siding. This belt is 75 feet long and travels at a speed of 175 feet per minute. Concentrates are scraped from it by wedge-shaped splitters and drop directly into the gondolas for shipment to the smelter. The weightometer is equipped with a cam mechanism which rings an electric bell for every 3.3 tons of product carried by the belt. This bell notifies the attendant to move the splitter so as to obtain an even distribution over the car. The concentrate conveyors have a life of about 78,000 tons of concentrates. They are driven by two 5-hp. motors which requir a power input of 2.47 and 1.65 kilowatts, respectiely.

The heavy machinery of the crushing plant is handled by crawls and manually operated chain blocks. Te rolls are served by a short crawl.

Repair parts for the remainder of the mill are handled with long slings and chain blocks suspended from the trusses or skidded into place.

Under the conditions noted, the follwing tabulation gives the number of men required and the time necessary to make repairs as listed.

Time and men required to make repairs

Nature of repair	Time required, hours	Number of men required
Crushers:		
Renewing jaw plates	32	6
No. 8 Traylor crusher:		
Babbitting eccentric	24	2
Installing eccentric	8 to 12	6
Putting concaves in barrel	24	2
Changing barrel	16	8
Mantle on shaft	8	6
Changing shaft	12 to 16	8
No. 6 K Gates crusher:		
Babbitting eccentric	8 to 12	2
Installing eccentric	6	6
Changing barrel	8	6
Mantle on shaft	8	6
Changing shaft	6	6
Rolls:		
Shrinking shells	12	5
Changing rolls	12 to 36	9
Babbitting roll bearings	24	6
Elevators:		
Punching belt	12	2
Installing buckets	12	7
Hum-mer screens:		
Changing one screen	3	3
Marcy mill:		
Installing feed end liners	8	5
Installing main liners	24 to 36	8
Installing grate bars	16	5
Hardinge mills:		
Renewing 20 pieces of liners and bars	8	5
Ball-tube mills:		
Renewal of feed end liners	6	4
Main liners	12 to 16	8
Screens	6	4
Minerals Separation machines:		
Changing impellers	16	4
Changing liners	16 to 20	6
Pumps:		
Changing liners	6	2
Changing liners and impellers	8	2
Filter:		
Changing cloth	12	6
Changing vacuum pipes	30	6

COSTS

Table 9 presents a summary of concentrator costs for the year 1929, and Table 10 gives concentrator costs in units of power and labor for the same period.

Table 9.--Summary of concentrator costs for the year 1929

	Operating				Repairs		Total
	Labor	Power[1]	Supplies	Reagents	Labor	Supplies	
Breaking	$0.0380	$0.0179	$0.0065	-	$0.0093	$0.0139	$0.0856
Intermediate crushing	0.0043	0.0094	0.0004	-	0.0037	0.0097	0.0275
Primary grinding	0.0203	0.0544	0.0469	-	0.0058	0.0254	0.1528
Secondary grinding	0.0173	0.0351	0.0318	-	0.0033	0.0063	0.0938
Classifying, screening, and conveying	0.0076	0.0064	0.0002	-	0.0110	0.0134	0.0386
Flotation	0.0224	0.0289	0.0002	$0.0329	0.0036	0.0052	0.0932
Dewatering and handling concentrates	0.0164	0.0035	0.0116	-	0.0010	0.0004	0.0329
Assaying and sampling	0.0029	0.0001	0.0081	-	0.0003	0.0003	0.0117
Tailings disposal	0.0317	0.0014	0.0051	0.0149	0.0046	0.0006	0.0583
Water supply	0.0013	0.0001	-	-	-	-	0.0014
Miscellaneous	0.0093	0.0006	0.0464	-	-	-	0.0563
Totals	$0.1715	$0.1578	$0.1572	$0.0478	$0.0426	$0.0752	$0.6521

1 - Cost per kilowatt-hour including power generated locally = $0.007909.

Table 10.--Summary of concentrator costs in units of labor and power

	Labor: Man shifts of 8 hours per ton of ore concentrated			Power	
	Operation	Repairs	Total	Kw. h. per ton	Per cent of total power
Breaking	0.00822	0.00189	0.01011	2.258	11.32
Intermediate crushing	0.00085	0.00076	0.00161	1.194	5.98
Primary grinding	0.00405	0.00118	0.00523	6.893	34.55
Secondary grinding	0.00346	0.00069	0.00415	4.441	22.26
Classifying, screening, and conveying	0.00151	0.00225	0.00376	0.811	4.07
Flotation	0.00427	0.00075	0.00502	3.652	18.30
Dewatering and handling concentrates	0.00376	0.00020	0.00396	0.442	2.21
Assaying and sampling	0.00061	0.00005	0.00066	-	-
Tailings disposal	0.00682	0.00093	0.00775	0.171	0.86
Water supply	0.00029	-	0.00029	-	-
Miscellaneous	0.00260	-	0.00260	0.088	0.45
Totals	0.03644	0.00870	0.04514	19.950	100.00

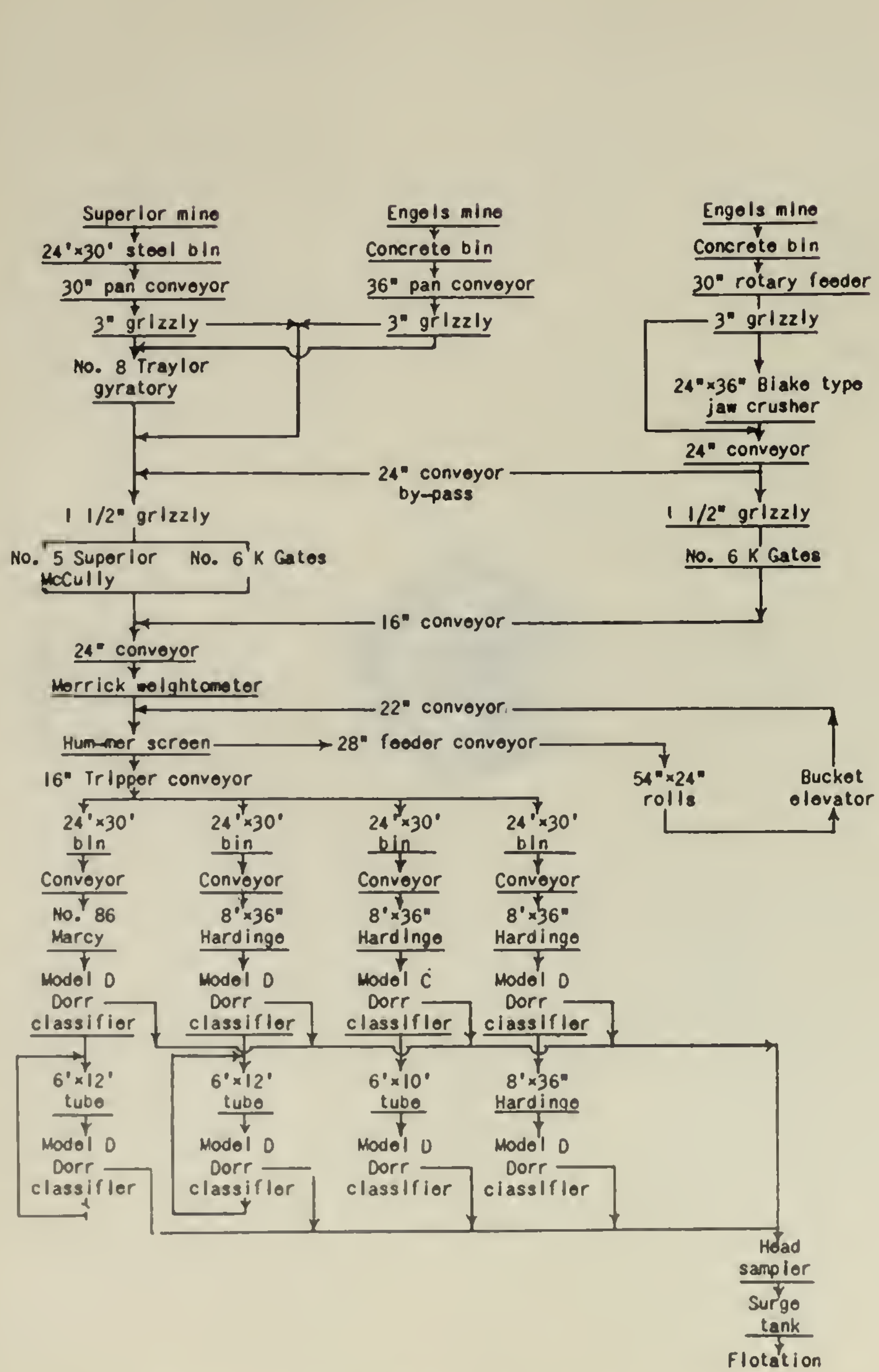

Figure 3.- Flow sheet of crushing and grinding operations

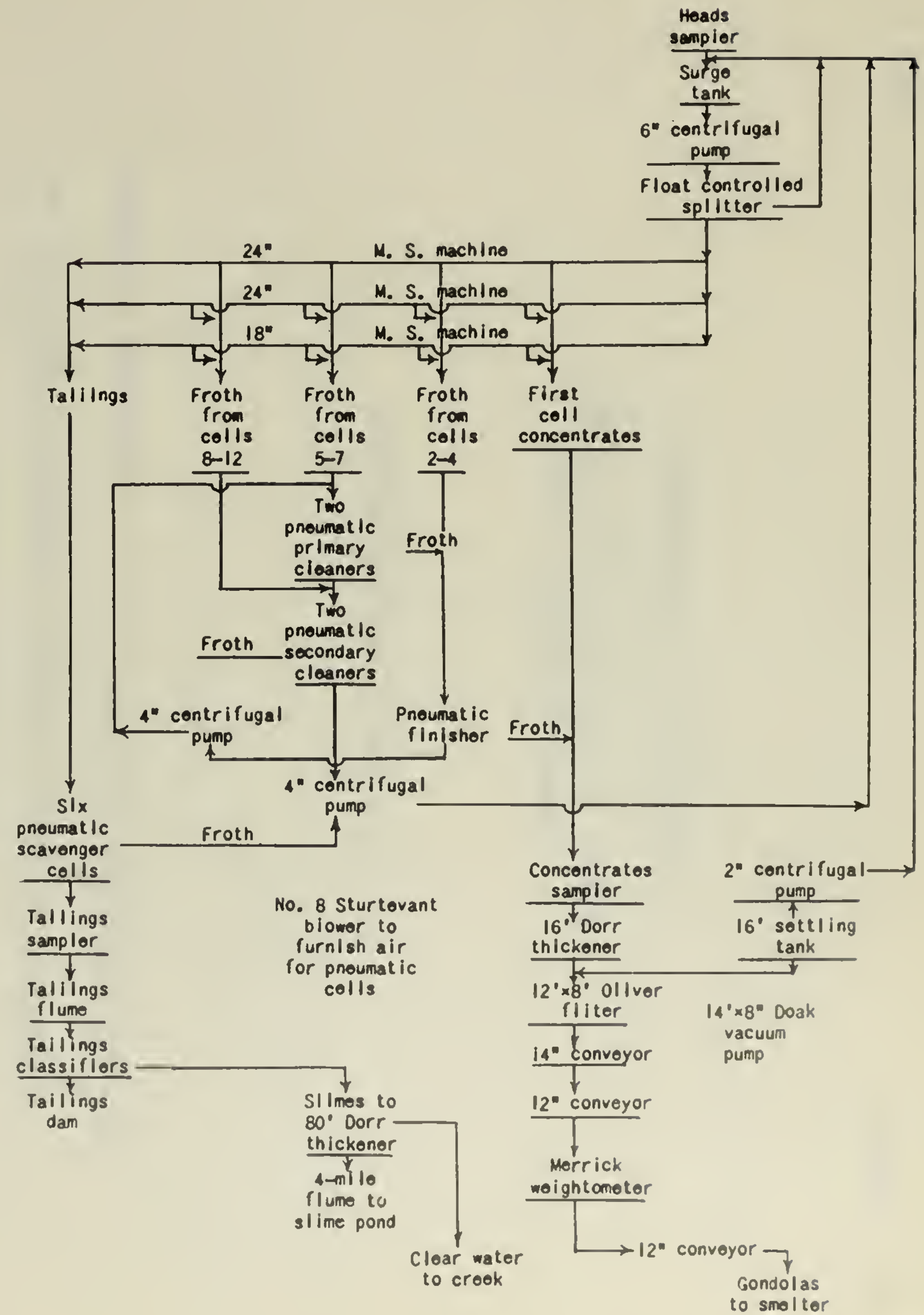

Figure 4.- Flow sheet of flotation and dewatering treatment

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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

MILLING METHODS AT THE QUESTA CONCENTRATOR OF THE MOLYBDENUM CORPORATION OF AMERICA, QUESTA, NEW MEXICO

BY

J. B. CARMAN

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January, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MILLING METHODS AT THE QUESTA CONCENTRATOR OF THE MOLYBDENUM CORPORATION OF AMERICA, QUESTA, N. MEX.[1]

By J. B. Carman[2]

INTRODUCTION

This paper, describing the concentrator of the Molygdenum Corporation of America in Taos County, northern New Mexico, 7 miles east of Questa, is one of a series on milling methods being prepared for and published by the United States Bureau of Mines. The mine from which the ore comes is 1¼ miles from the mill, and produces about 40 tons of ore per day by the cut-and-fill method.[3] The ore contains about 5 per cent of MoS_2. The mill has a capacity of 50 tons of ore per day and utilizes flotation methods. Water is taken from Red River. Power is generated by a water wheel, or by a Diesel engine in times of water shortage during the winter. The mill is located close to the bottom of a canyon, in order to obtain maximum head on the water wheel, and the flow of ore is therefore maintained partly by gravity and partly by an elevator and pumps.

ORE TREATED

The MoS_2 content of the concentrator feed ranges from 4.5 to 7.5 per cent. The gangue minerals are quartz, pyrite, altered feldspar, biotite, fluorite, small amounts of rhodochrosite and calcite, and the minerals of the granodiorite country rock.

The molybdenite is not disseminated but occurs as fissure fillings and in brecciated zones of the granodiorite. A portion of it, however, is very intimately associated with quartz or pyrite, and in this form is not readily separated by grinding. The coarse ore has numerous joints or fracture surfaces which are coated with thin black films of molybdenite. Molybdenite of this character can not be split from the rock and this strong mechanical attachment apparently persists even through the grinding process.

The ore in general breaks and grinds readily and the molybdenite is very friable. The run-of-mine ore is 8-inch maximum size and 75 per cent of it will pass through a bar grizzly with 1½-inch spaces. The moisture content of the oro averages 3 per cent.

HISTORY OF OPERATIONS

The first mill was built in 1923, and as originally operated, contained eight 4-foot, Brown pneumatic, matless-type, flotation cells. The froths from the first one to three cells were finished concentrates which contained a maximum of 66 per cent of MoS_2. The froths from the remaining cells were returned to the head cell as middlings.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6551."

2 One of the consulting engineers, U. S. Bureau of Mines, and manager, Questa property, Molybdenum Corporation of America.

3 Carman, J. B., Mining Methods of the Molybdenum Corporation of America at Questa, New Mexico: Information Circular, Bureau of Mines, in press.

Three Ruth flotation cells, arranged in parallel, were later installed as cleaners to increase the grade of the finished concentrates. After these cleaner units started to operate, difficulty developed because of the accumulating of slimes in the flotation circuit. The addition of locally made flotation cells for additional cleaning of concentrates further increased the grade of concentrates, but this arrangement seriously increased the amount of middling returns to the head of the flotation circuit and further increased the difficulty due to building up of slimes in the circuit. Three additional Brown flotation cells were then installed for retreating middlings. These cells were later operated as scavenger cells, following the 8-cell rougher machine, and one additional Brown cell was installed to replace them for middlings retreatment.

During 1928 and 1929, preliminary to the construction of a new mill, tests were conducted and experimental changes were made in the flow sheet. A description of this experimental work given in chronological order follows.

Acting on the results of screen assay tests which indicated that concentrator losses occurred in minus 200-mesh sands, the grinding practice was changed to yield finer grinding. The flotation of the finer ground product yielded a higher recovery of molybdenum but a lower grade of concentrates. These tests indicated that net results favored the coarser grinding.

A test was conducted to determine the recovery made by each cell of the Brown 8-cell rougher and the Brown 3-cell scavenger machines. The results of this test indicated that the low solid contents in the pulps treated, caused by diluting water returned to the circuit with middlings, was responsible for low recoveries in the last six cells. An 8 by 14 foot Dorr thickener was added to the circuit to remedy this condition. The treatment of pulps, which contained a higher percentage of solids, reduced the loss of molybdenum in the flotation tailings about 50 per cent. The thickener overflow, however, became muddy after 10 days of operation. This was believed to be due to insufficient thickener capacity, and the test was abandoned.

Three Callow cells were installed to replace the 3-cell Ruth cleaner unit. This change resulted in slightly increased losses of molybdenum in the plus 200-mesh tailings sizes and slightly decreased losses in the minus 200-mesh sizes.

Baffles were added to the Brown rougher machine for the purpose of increasing the percentage of plus 200-mesh MoS_2 floated in the rougher cells. This practice, however, was discontinued.

Sideboards were added to the overflow lips of the Callow cleaners for the purpose of forcing the froth to travel to the heads of the machines for discharge and thus drop some of the gangue. This method of operation improved the grade of concentrates but at the expense of higher losses in the tailings.

Thickening of the middlings pulps before returning them to the head of the flotation circuit was again tested, but this time by the installation of a larger thickener than was at first used for this purpose. The test was not successful, due to the accumulation of colloidal material in the circuit.

The thickened middlings pulp was then returned to the classifier of the grinding unit. The classifier rake speed was decreased and the slope increased for the purpose of preventing the return of fines to the grinding mill. This method of handling middlings returned the slimes to the rougher flotation unit and was abandoned because it produced "wild froth" in the rougher cells which in turn lowered the grade of concentrates without decreasing the loss in the tailings.

The addition of 0.10 pound of caustic soda per ton of ore treated to No. 1 cleaner cell aided the control of froth which had been difficult in the cleaner units due to accumulation of slimes. This reagent increased the grade of cleaner concentrates from 3 to 4 per cent

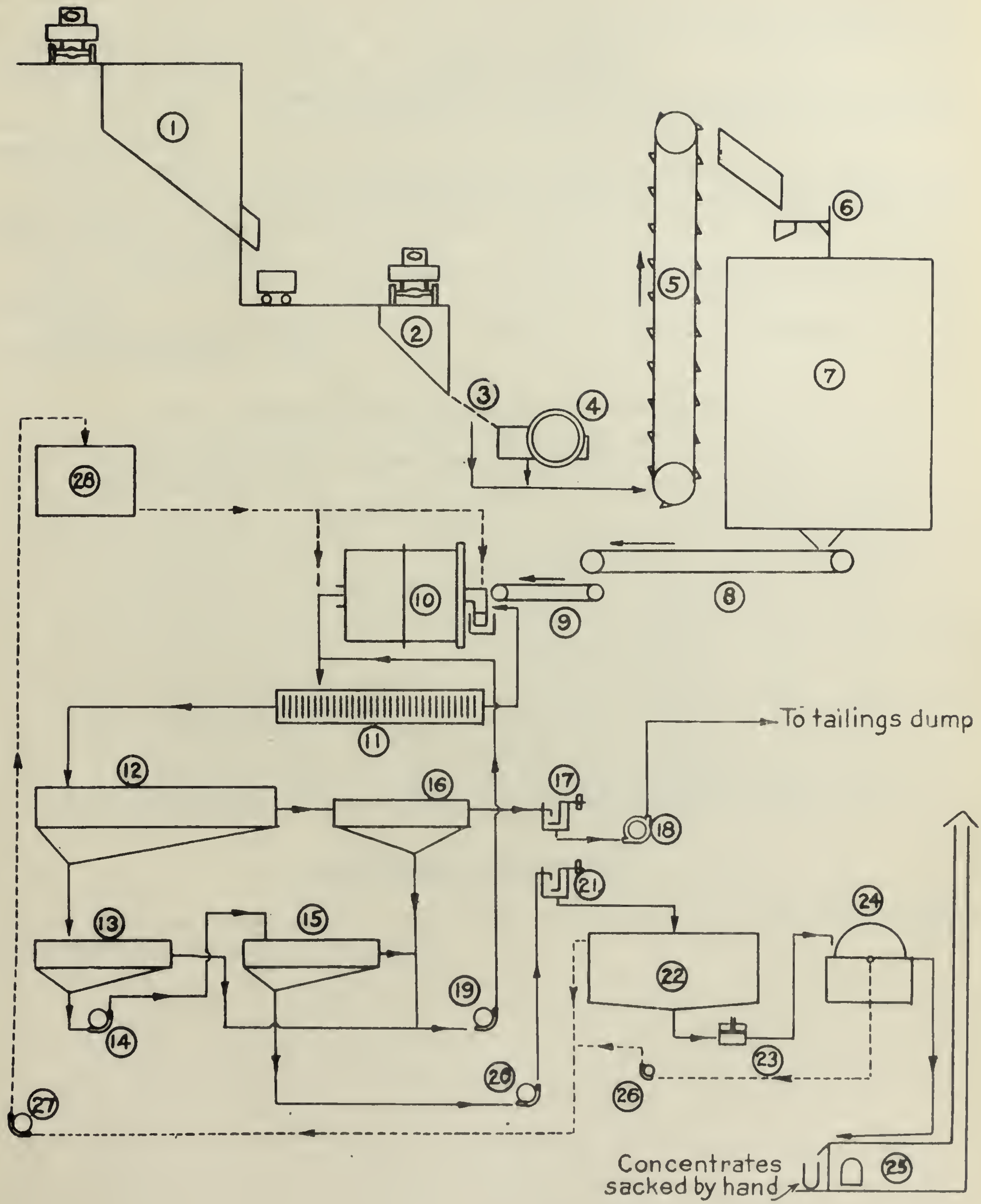


Figure 1.- Flow sheet of ore treatment

and decreased the loss of MoS_2 in the cleaner middlings by 10 per cent. The caustic soda, however, finally began to appear in the mill feed water and this condition increased the tailings loss in the rougher cells. After reducing the amount of caustic soda added to 0.05 pound per ton of material treated, it was finally discontinued.

The classifying of flotation concentrates, with the idea of separating the coarser material for additional grinding, indicated that higher-grade concentrates could not be produced by this method.

The No. 1 cleaner cell was discontinued as a cleaner and was put into service as a primary rougher machine. This change resulted in decreased loss in tailings of 1 to 1.5 per cent, increase of 5 to 6 per cent in the grade of concentrates and an increase in recovery from 88 to 93 per cent.

Table 1 presents metallurgical data during this period of experimental work in the old concentrator.

Construction of the new concentrator was started in 1929 and the operation of this new plant began on February 1, 1930.

Table 1.- Metallurgical data for experimental periods in the old concentrator

Test period	Reagents used per ton of ore treated, pounds: Pine oils: Yarmor	Pensacola No. 100	G.N.S. No. 5	Total	Caustic soda	Sodium cyanide	Xanthate	Assays per cent MoS_2: Heads	Concentrates	Tailings	Recovery of MoS_2, per cent	Plus 100-mesh material in heads, per cent	Solids in flotation pulp, per cent
1	3.00	3.00	-	6.00	0.09	0.20	-	5.40	73.35	0.58	89.64	4.00	20
2	-	1.60	0.40	2.00	0.09	0.20	-	4.84	72.42	0.46	89.73	1.50	20
3	1.375	1.375	-	2.75	0.09	0.20	-	6.33	71.96	0.68	89.26	3.00	19
4	-	1.65	0.85	2.50	-	0.18	-	6.10	72.50	0.86	86.55	0.70	18
5	-	1.60	0.40	2.00	-	0.16	-	6.19	75.35	0.74	88.30	0.85	18.5
6	-	1.25	0.25	1.50	-	0.14	-	6.30	74.48	0.67	89.36	2.05	20
7	-	0.83	0.17	1.00	-	0.09	-	5.28	75.53	0.78	85.23	2.67	22.5
8	-	0.58	0.12	0.70	-	0.09	0.16	5.70	74.30	0.82	85.64	-	-
9	-	0.83	0.17	1.00	-	0.09	-	5.19	74.50	0.64	87.66	2.00	21

PRESENT METHOD OF CONCENTRATING

The flow sheet of ore treatment as practiced at present is given in Figure 1; the sizes of equipment used are shown in the legend.

Legend for Figure 1

Number[1]	Machines
1	Reserve coarse-ore storage bin, 300-ton capacity.
2	Coarse-ore bin, 12-ton capacity.
3	Bar grizzly, 1½ inch openings.
4	Universal jaw crusher, 9 by 14 inch.
5	Bucket elevator.
6	Mechanical sampler.
7	Fine-ore storage bin, 100-ton capacity.
8	Belt conveyor, 20-inch.
9	Belt conveyor, 12-inch.
10	Marcy ball mill, No. 54.
11	Dorr simplex classifier, 3 by 18 foot.
12	Callow rougher flotation machine, 2 by 21 foot.
13	Callow primary cleaner, 16-inch by 8-foot.
14	American centrifugal pump, 2-inch.
15	Callow secondary cleaner, 16-inch by 8-foot.
16	Callow scavenger, 16-inch by 8-foot.
17	Mechanical sampler.
18	Wilfley sand pump, 2-inch.
19	American centrifugal pump, 2-inch
20	American centrifugal pump, 2-inch.
21	Mechanical sampler.
22	Dorr thickener, 14 by 8 foot.
23	Diaphram pump.
24	American filter, 4-foot, single leaf.
25	Stove for drying concentrates.
26	Cameron centrifugal pump, 1-inch.
27	American centrifugal pump, 2-inch.
28	Storage tank for mill water.

1 Numbers correspond to numbers given in Figure 1.

Coarse Crushing

Ore is transported from the mine to the concentrator in automobile trucks during the day shift only. At the concentrator, the trucks dump the ore into a 12-ton capacity bin which serves the crusher. If this small bin is full, the ore is delivered to a 300-ton capacity storage bin from which it is transported to the crusher bin in a 1-ton capacity mine car.

The Universal 9 by 14 inch jaw crusher is driven by belt and is set at 1½-inch openings. A bar grizzly, with 1½-inch openings, is placed between the bin and crusher. The crusher has capacity to handle 60 tons of Questa ore per eight hours but crushes, on the average, about 16 tons per eight hours. The crusher product contains about 5 per cent plus 1½-inch ring size and about 40 per cent minus ½-inch ring size. It joins the undersize of the grizzly and is delivered to a 100-ton capacity fine-ore bin by a bucket elevator.

Grinding

From the fine-ore bin the ore is delivered to a 20-inch conveyor belt, feeding a 12-inch conveyor belt which in turn feeds a No. 54 Marcy ball mill. The ball mill which operates in closed circuit with a Dorr rake classifier is driven at a speed of 27 r.p.m. through a friction clutch and is equipped with a grate discharge which has 3/16-inch openings.

The ball load is maintained at about 4,200 pounds by the addition of 4-inch forged steel balls. Ball consumption amounts to about 1½ pounds per ton of ore ground.

Liners are of molybdenum-chrome steel, and liner consumption amounts to 0.934 pound per ton of ore ground.

During the first ten months of operation in 1930 the Marcy mill operated 6,878 hours out of a possible 6,936 hours. During this period, one set of side liners was replaced and various construction difficulties were encountered, but the lost time amounted to less than 1 per cent.

The 3 by 18 foot Dorr classifier which operates in closed circuit with the Marcy mill is set at a slope of 2¾ inches to the foot, and the rakes have a speed of 12 strokes per minute.

The launder used to convey the ball-mill discharge to the classifier has a slope of 1½ inches per foot and is lined with 3/16-inch steel plate. The launder which returns classifier sands to the ball mill has a slope of 3¾ inches per foot; this slope is the maximum that could be obtained with the set-up that existed at the time of rebuilding the mill.

Pulp density in the ball mill is maintained at about 80 per cent of solids and that in the classifier from 20 to 22 per cent of solids. The classifier is equipped with a hydrometer, installed at the overflow end, to assist in maintaining the desired pulp density. The hydrometer operates a pointer which indicates the percentage of solids in the classifier on a dial. The dial is visible from various distant points of the building. This arrangement has proved to be of great value in indicating quickly such conditions as changes in water or ore feed or a change in the character of the ore.

The Marcy mill is at present grinding about 39 tons of ore per 24 hours and the classifier is handling a circulating load of about 280 per cent based on new ore feed. Both machines are operating far below their capacities.

Table 2 presents typical screen analyses of final and intermediate grinding products and also indicates the percentage of solids in the various pulps.

Table 2.- Typical screen analyses of final and intermediate grinding products

	Test 1			Test 2			Test 3		
	Ball mill discharge, per cent	Classifier sand, per cent	Classifier overflow, per cent	Ball mill discharge, per cent	Classifier sand, per cent	Classifier overflow, per cent	Ball mill discharge, per cent	Classifier sand, per cent	Classifier overflow, per cent
Solids in pulp	80.0	70.00	20.5	85.0	68.0	21.0	77.9	70.0	20.0
Screen sizes, mesh:									
Plus 10	1.33	1.72	-	1.49	1.92	-	1.91	1.85	-
Minus 10 plus 20	3.60	3.69	-	3.48	6.75	-	6.18	3.26	-
Minus 20 plus 30	7.11	7.61	-	6.60	7.75	-	6.98	5.74	-
Minus 30 plus 40	8.76	8.75	-	7.60	10.05	-	7.38	8.56	-
Minus 40 plus 100	44.89	48.05	0.85	40.00	45.40	0.51	35.92	45.05	0.68
Minus 100 plus 150	12.76	14.00	9.48	14.55	14.01	7.88	14.45	17.05	9.87
Minus 150 plus 200	11.30	10.01	20.80	13.00	8.11	17.38	12.90	11.65	16.44
Minus 200	10.01	6.56	68.38	12.65	5.16	73.49	13.28	6.60	72.01
Total	99.76	100.39	99.51	99.37	99.15	99.26	99.00	99.76	99.00

Flotation

The classifier overflow pulp is treated in a Callow rougher machine. The rougher produces concentrates which are cleaned in two stages of callow cells. The tailings of the rougher are treated in a Callow scavenger cell which produces final waste tailings. The middlings products from the primary and secondary cleaner cells together with the froth product of the scavenger cell are returned to the head of the flotation circuit.

The Callow rougher is a 7-pan 2 by 21 foot machine. The cleaner and scavenger units are 4-pan 16-inch by 8-foot cells. These cells are equipped with 4-ply, $\frac{3}{4}$-inch square stitched canvas blankets. The blankets of the rougher and scavenger cells have a life of about six months and those of the cleaner cells last somewhat longer.

Air is supplied to all flotation cells at a pressure of 4 to $4\frac{1}{2}$ pounds per square inch by a No. 2 belt-driven Root blower. The volume of air consumed amounts to about 800 to 900 cubic feet per minute which is equivalent to about 8 cubic feet of free air per square foot of blanket area per minute. The blower is at present located in the mill building, but it is planned to move the blower to a position outside the building because dust in the air accumulates on the rotors and increases friction losses. Forced oil feed is required to lubricate the bearings, which tend to become gummy.

The flotation reagents used comprise Pensacola No. 100 pine oil, used as a collector; G. N. S. No. 5 pine oil, used as a frother; and sodium cyanide, used to depress pyrite. The pine oils are mixed in the proportion of 5 parts of Pensacola to 1 part of the G. N. S., by volume, and the mixture is added to the ball mill scoop box at the rate of 1 pound per ton of ore. The sodium cyanide is added to the secondary cleaner as a 10 per cent solution at the rate of 0.08 pound per ton of ore by a disk and cup type feeder.

A sketch of the feeder used for the pine-oil mixture is presented in Figure 2. A board is caused to slide vertically up and down in wooden guides by means of a crank arm fastened to a speed reducer and driven by a belt from the same counter shaft which drives the ball-mill feed conveyor. A small cylindrical cup is attached to the lower end of the board by means of metal strips which form a fork. The lower end of the cup is fastened to a wire pivot which is held by the forked strips. A length of heavy wire is welded to the wire pivot and slides over a guide pin which is attached to the framework of the device by a metal strip. As the cup goes down, the wire which slides over the guide pin causes it to assume an erect position, and it is filled with the liquid into which it dips. As it rises on the upstroke above the edge of the pan holding the reagent, the wire slides forward over the guide pin and causes the cup to tip forward. According to the position of the guide pin and the shape of the slide wire, the cup tips more or less, and a part of its content is emptied onto the lip on the edge of the pan and flows by gravity through a small pipe to the ball-mill scoop box.

The principal reasons for the use of pneumatic flotation machines at this plant are low power requirements, simplicity of installation and upkeep, and ease of operation as compared to machines of the mechanical agitation type. It was, moreover, known that they would give satisfactory differentiation between the molybdenite and the pyrite, for the pyrite in the ore appears to be slightly tarnished as it comes from the mine and is therefore readily depressed.

Sideboards are used on the two cleaner machines to force the froth to travel from the lower end to nearly the head end before being discharged. This method of operating gives cleaner concentrates, because of the increased opportunity to drop gangue and middlings particles.

Metallurgical data of the flotation circuit is given in Table 3.

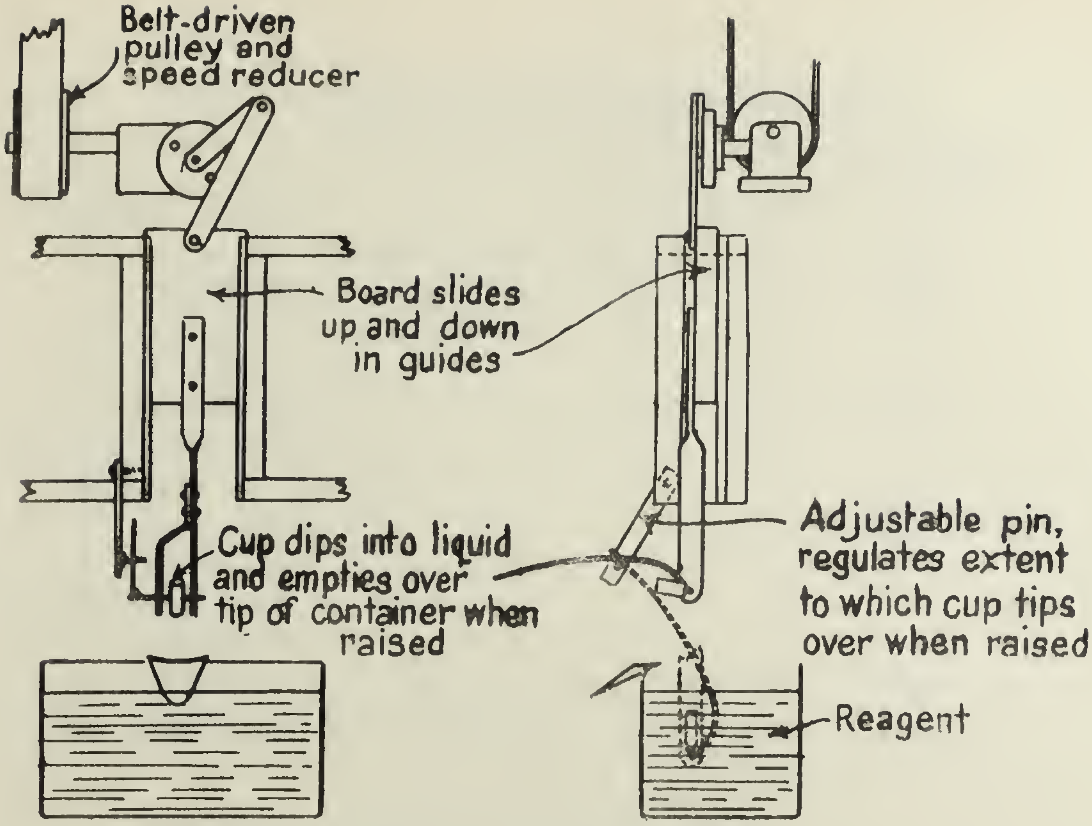


Figure 2.- Flotation-reagent feeder

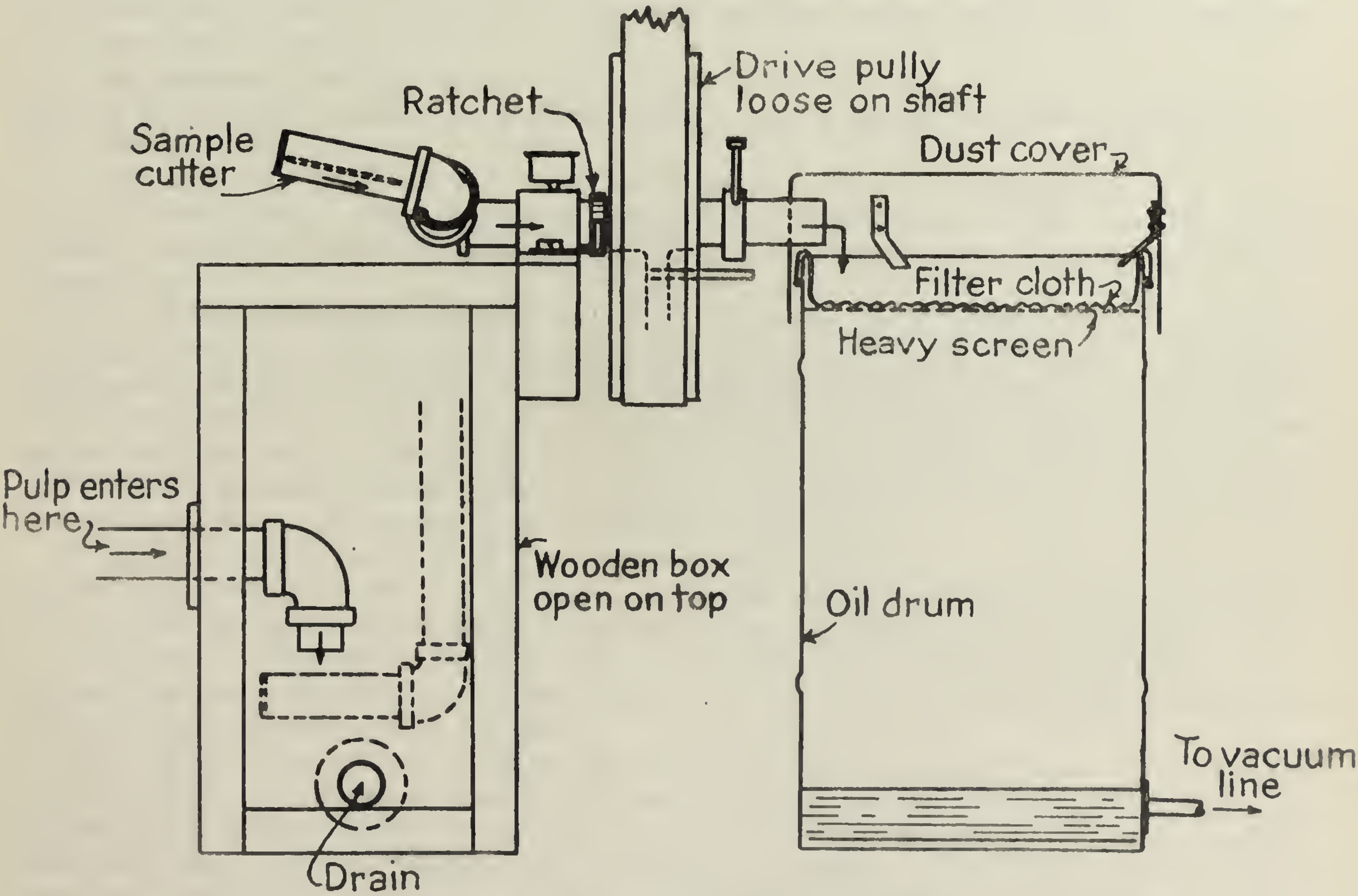


Figure 3.- Automatic sampler

Table 3.- Flotation circuit metallurgical data

	MoS_2, per cent
Rougher cell:	
Heads, not including middlings returned	5.2
Concentrates	30.0
Tailings	0.9
Primary cleaner cell:	
Concentrates	55.0
Middlings	4.0
Secondary cleaner cell:	
Concentrates	74.5
Middlings	7.0
Scavenger cell:	
Concentrates	1 to 3
Tailings	0.6
Recovery in final concentrates	87.7

DEWATERING AND HANDLING OF CONCENTRATES

The finished concentrates pulp of the secondary cleaner cell contains about 3 per cent of solids and is pumped to a 14 by 8 foot Dorr thickener, which operates with a rake speed of 1 revolution in eight minutes.

The thickener underflow pulp contains about 25 per cent of solids and is further dewatered in a 4-foot, single-leaf American filter. The thickener overflow is pumped to the mill storage tank.

Vacuum at the filter is maintained at about 17 inches of mercury by an 8 by 5 inch vertical vacuum pump. Palma reinforced canvas-twill filter cloth is used; the life of the cloth is about two weeks. Compressed air for the filter "blow" is taken from the Root blower line.

The filter cake which contains from 10 to 12 per cent of moisture drops by gravity through a chute to a shelf located adjacent to the drying stove. This stove is a rectangular brick-walled inclosure, 4½ by 6½ feet inside dimensions by about 3 feet high, and is equipped with a cast-iron top. Soft coal is burned on a grate, the hot gases passing to the other end of the dryer and up a metal chimney. The concentrates are pulled by hoe onto the top of the dryer at the cool end and worked over as they are gradually moved toward the hot end. The moisture content is reduced to an average of 3½ per cent.

The only cost connected with drying is for fuel, as the man who sacks and weighs the concentrates tends the dryer. The fuel consumption amounts to about 6 tons of coal per month, which is equivalent to about 1½ pounds of coal per pound of water eliminated.

The dryer, particularly the smoke pipe, performs the additional duty of heating the mill through the cold winter months, and some part of the coal used for drying concentrates would be burned for this purpose in any case.

The drier concentrates are pulled off the dryer by hoe into sacks, each of which holds about 80 pounds. In sacking, some dust is stirred up; a collector hood is being installed to avoid this nuisance and to prevent loss of concentrates. The sacks are of burlap and are lined with paper which is cemented to the cloth with asphalt. These sacks cost 14 cents each, and although they constitute one of the largest supply costs for the mill they are very satisfactory. Heavy jute sacks with canvas liners were tried, but they weighed more and the

loss in and through the fabric was considerable, in spite of a thorough dusting out at the receiving end. The sacks are not reused because of their loss of strength. They are burned to recover the concentrates which can not be removed by shaking.

The sacks are closed by being sewed with twine, after which they are weighed on platform scales, tagged, and stored for shipment.

DISPOSAL OF TAILINGS

The tailings pulp from the scavenger flotation cell is pumped to the tailings pond by a 2-inch Wilfley sand pump. The pipe line discharges into a wooden launder, 4 by 4 inches inside dimensions, located at the tailings dam. This launder is set at a slope of $\frac{3}{8}$ inch per foot around the edge of the dam. Holes are bored at regular intervals in the bottom of this launder to permit the tailings to drop into a short distributing launder. The later has notched edges to spread the material evenly as possible and thus prevent its piling up; it is moved from point to point along the dam as desired. A low wall is heaped up, by shoveling, a few feet outside the line of the main launder to retain the tailings. The clear water flows through a stand pipe located in the middle of the pond and is discharged into the river.

WEIGHTING, SAMPLING, AND ASSAYING

The concentrator heads are sampled as the ore stream is discharged into the fine-ore bin. A mechanical sampler, which consists of a box slowly rotated on the end of an arm 3 feet long, cuts the ore stream and removes 4 per cent of the heads as a sample. The box, after cutting the ore stream, is discharged by a tripper which opens the counterweighted bottom. The 24-hour sample is reduced to about 500 pounds by shoveling. This heads sample has been found to give inaccurate and erratic results. The error apparently increases with the moisture content of the ore and is found to be a minimum when the moisture content of the heads is around 2½ per cent. The heads sample is used for determinations of molybdenum and moisture contents of the concentrator feed.

The tailings sample is obtained by a mechanical sampler located between the scavenger flotation cell and the Wilfley sand pump. Referring to Figure 3, a pulley which is driven by belt is mounted loosely between collars on a length of pipe. One end of the pipe passes through a bearing which in turn is mounted on the side of the sample box, and continues through a 90° elbow, another length of pipe, and a second elbow, to the cutter itself. The latter consists of a length of 2-inch pipe about 6 inches long, closed at the far end and having a narrow slot cut in the side which is uppermost when the sample pipe is in the position shown by the dotted lines. The sample consists of the amount of pulp which enters the slot as the pipe passes through the stream. As the pulley turns slowly, a stud which is set in one of its spokes strikes a long set screw attached to a collar on the shaft pipe, causing it to turn. When the sample pipe just passes its highest point of travel, gravity causes it to fall swiftly, the sample being cut at the lowest point of travel as the cutter passes through the stream of pulp. The momentum of the falling cutter causes it to pass more than halfway up the other half of a complete revolution. A ratchet and spring arrangement on the shaft-pipe prevents it from falling backward, and holds it in such a position that the sample drains out through the radial arm and the shaft into a filter. The latter is made from an oil drum, by removing the head, tapping a pipe into the bottom for connection to the mill vacuum line, and attaching a screen near the top of the drum. A muslin filter cloth fastened to a snug-fitting wire hoop is put over the top of the drum and rests on the screen. a loose-fitting cover is provided, slotted at one side to pass over the pipe shaft of the sampler. The filtrate water, which is perfectly clear, is discharged intermittently from the

drum, as required, by opening a valve in the suction line. Filter paper was formerly used in place of muslin but was found to be more expensive, as it could be used only once, whereas the muslin is washed and reused.

The 8-hour sample which amounts to about 1/900 cut weighs about 30 pounds. It is discharged from the filter into a tub and dried on a stove. The tailings sampler has been found to produce accurate samples.

A similar sampler is used to take shift samples of the concentrates from the secondary flotation cleaner cell. The 8-hour concentrates sample weighs about 30 pounds and is used only for operating control

All ore is weighed in the trucks before being dumped into the mill bins. The weight of fine ore sent to the ball mill is determined by collecting the discharge from the conveyor belt at ½-hour intervals, for a known length of time. All such samples are weighed. and the weights are averaged for each shift. The weight as determined by this method usually agrees with the truck weighings within 1 ton per month.

All concentrator samples are assayed by a volumetric method, which follows, in the main, the procedure described by J. P. Bonardi in United States Bureau of Mines Bulletin 212, Analytical Methods for Certain Metals, page 109. A weighed portion of the sample is fused with sodium peroxide, dissolved in water, and the solution filtered. A Jones reductor is used to reduce the molybdenum, after which titration is performed with a potassium permanganate soution which has been standardized against sodium oxalate obtained from the Bureau of Standards. All results are expressed in percentage by weight of MoS_2. About half the time of one man is devoted to assaying the mine and mill samples.

WATER SUPPLY

An ample supply of water for milling and camp use is available in Red River. The minimum creek flow is about 10 second-feet, which is more than ample for mill requirements.

The fresh water supply may be taken from the pipe line which serves the Pelton wheel or directly from the river bottom above the mill through a pipe line which discharges by gravity into a sump in the mill. Water for camp use is elevated to the camp water-supply tank from this sump by a Goulds triplex pump.

The water is relatively soft and does not contain impurities harmful to milling operations except during intervals in spring and early summer when the river water is muddy enough to interfere with flotation operations. At such times water for milling purposes is obtained from a small stream leading into Red River directly across from th mill. This water flows by gravity through a pipe line which crosses the river and connects with the mill supply system

The new water consumption amounts to about 2½ tons per ton of ore milled. An approximately equal amount is returned to the ball-mill circuit in the middlings pulp and some is reclaimed by the thickener and filter.

The thickener overflow water and the water from the filter are collected in a small sump and lifted by a 2-inch American centrifugal pump to the mill water-supply tank located at the head of the mill.

The camp uses about 6 gallons of water per minute from the camp supply tank. Probably not over 150 to 175 persons out of a total of 250 use this supply.

POWER

Power for milling operations is furnished by a water wheel, except in winter months when the water is low. During the low-water period a Diesel engine provides the power.

The water wheel was installed when the mill was built in 1923, and at that time sufficient water was available for continuous operations on the basis of 20 tons daily capacity. When the mill was rebuilt in 1930, however, it was known that sufficient water power would not be available during the entire year, and the engine was installed at that time.

Water for the operation of the wheel is diverted from Red River by a small, cribbed, rock-filled dam, which is about 20 feet wide and 4 feet high; it flows through a wooden flume, a distance of 6,800 feet to the penstock above the wheel.

The water drops from the penstock to the wheel through two 16-inch diameter straight-riveted, flanged pipes, which join in a short length of 20-inch diameter pipe at the bottom. The second pipe was added when the mill was rebuilt. The intake pipes are 400 feet long and the head on the wheel is 110 feet.

The power element is a Pelton water wheel which has a pitch diameter of 66¼ inches and which is equipped with a 5 7/16-inch shaft. This wheel was originally designed to deliver about 350 hp., and was obtained secondhand for this installation. Its speed under the present normal load is 146 r.p.m. The original 4¼-inch nozzle was replaced by a double nozzle having one 3-inch and one 4-inch opening.

The wheel is connected by belt to the main drive shaft of the mill through a jaw clutch. A 15-kilowatt, 110-volt, direct-current generator, connected to the wheel by belt, supplies power for electric lighting. A 24-inch Pelton wheel drives this generator when the larger wheel is not in operation. A third and still smaller wheel furnishes power for the machine shop.

The construction of the flume which is used at present for conveying water to the Pelton wheels is shown in Figure 4. This flume normally carried about 15 inches of water at a velocity of 4 feet per second, a flow which amounts to approximately 14 second-feet. It is set on a grade of 1½ feet per thousand feet and was built at a cost of $2.50 per foot. It is equipped with a sand box located about 200 feet from the upper end which effectively prevents the accumulation of sediment in the flume.

The engine is a 4-cycle, 4 cylinder, Chicago-Pneumatic, 10¾ by 13¾ full Diesel type. It is rated at 160 hp. at sea level and delivers 119 hp. at its present altitude of 8,100 feet. The engine is equipped with a Midwest intake air filter, a De Laval oil separator which removes considerable carbon from the lubricating oil, and an exhaust muffler pot which very successfully eliminates the noise of the exhaust. Cooling water on leaving the engine goes to a riffle-type cooler on the roof of the building and from there to a 3,000-gallon capacity water tank. Starting air at 350 pounds pressure is furnished by a 2-stage vertical 4½ by 2 by 4½ inch Gardner-Denver "Rix" compressor, which is connected by belt to a 5 to 8 hp., 1-cylinder, New-way air-cooled gasoline engine. The Diesel engine is connected by belt to the main shaft through a friction clutch. A jaw clutch in the main shaft disconnects the Diesel pulley when the engine is idle.

Comparative operating costs for power generated by the Pelton water wheel and by the Diesel engine are presented in the tabulation which follows.

Operating costs for power generated by the Pelton water wheel and the Diesel engine

Pelton water wheel	Cost per hp. hour	Diesel engine	Cost per hp.-hour
Operating labor....	$0.000833	Fuel oil	$0.00738
		Lubricating oil	0.00041
			$0.00779

These operating costs are based on the following data: Operating labor for the water wheel consists of part of the time of one man to inspect and repair the flume. Operating labor is not charged to the engine, as an increase in force does not result from its use. The cost of operating and repair supplies for the wheel is negligible. The engine consumes 125 gallons, or 815 pounds of fuel oil per 24 hours at an average three-quarters load, at 1.845 cents per pound. It also uses 0.0546 gallon of lubricating oil per hour at 75 cents per gallon. The average output of both prime movers is estimated at 100 hp.

The cost of installing the Pelton wheel and flume line was about $23,000. That of the Diesel engine and all accessories, including the building and foundations, was about $15,000. Repair costs on the Diesel during its first year of operation have been negligible, and offer no basis on which to estimate this item for the future. Repair costs on the flume will doubtless increase. The saving in power cost by the use of available water power amounts to over $500 per month. The operating cost for Diesel power at this plant is ten times that of water power.

CONVEYING AND PUMPING

A bucket elevator receives the broken ore and by-passed fines below the jaw crusher and elevates these products about 22 feet to the top of the fine-ore bin.

A 24-inch conveyor belt, 15 feet long, delivers the ore from the fine-ore bin to a 12-inch cross conveyor belt, which in turn feeds the ball mill. The 24-inch conveyor is driven intermittently by a ratchet and pawl feeder, and the speed of this belt, which averages about 20 inches per minute, governs the rate of feed to the ball mill.

The classifier overflow pulp is conveyed by launder to the rougher flotation machine. The concentrates froth from the rougher flows by launder to the primary cleaner. The concentrates froth from the primary cleaner cell is pumped to the secondary cleaner and the froth from that machine is pumped to the thickener by 2-inch American centrifugal pumps. The rougher-cell tailings are conveyed by launder to the scavenger machine. The middlings of the cleaner cells and the froth from the scavenger machine are collected in a sump and returned to the ball-mill discharge by a 2-inch American centrifugal pump. Tailings from the scavenger cell are elevated about 25 feet by a 2-inch Wilfley sand pump and discharged into a 2-inch pipe line which extends on a down grade to the tailings pond.

The pumps mentioned are each driven by belt. The launders, 8 inches wide and 6 inches deep inside, are made of 1-inch lumber and placed on a slope of about $\frac{1}{4}$ inch per foot, with the exception of the froth launder from the rougher to the primary cleaner, which has a slope of about $\frac{3}{4}$ inch per foot. The launders are not lined.

CONCENTRATOR LOSSES

From the results of microscopic analyses, made prior to 1930, using magnifications of 30 diameters, it was concluded that grinding to minus 150-mesh size liberated practically all of the MoS_2 from the gangue. At this time, therefore, it was believed that all losses of MoS_2 in tailings were due to the presence of slimes contained in the flotation pulp.

The ore contains some primary slimes but probably not in sufficient quantity to interfere with flotation results, and it was therefore concluded that the losses were occasioned by slimes produced in grinding operations.

The installation of a grate-discharge ball mill decreased the amount of slimes produced in grinding operations, but large amounts are still produced and efforts have been made to further decrease the slime content of the flotation pulp.

At times the concentrator feed has contained large amounts of kaolin or other talcose alteration products, but the removal and separate treatment of such material has never been considered because of the small amounts present and the assumption of increased operating costs for such procedure.

Early in 1930 a microscope capable of magnifying 250 diameters was acquired, and its use revealed that the MoS_2 was not liberated as completely as was formerly believed. Microscopic analyses indicated that from 70 to 80 per cent of the MoS_2 contained in the tailings was attached material.

Particles of quartz and pyrite which had appeared perfectly barren at magnifications of 30 diameters were seen to have numerous extremely small particles of molybdenite attached to them. The microscope likewise showed that 60 to 70 per cent of the gangue contained in the concentrates was attached to molybdenum sulphide. It was evident that crushing sufficiently fine to liberate such material was impracticable, and at the same time it was concluded that slimes contained in the flotation circuit had not been as serious a cause of loss as was formerly believed.

Experimental work is being carried on with the object of recovering more of the attached MoS_2, which is at present lost in the tailings. If the flotation of these included grains proves feasible the grade of concentrates can be improved by additional grinding and flotation of the middlings product obtained in a separate unit.

LABOR

The mill crew consists of one man at the crusher during day-shift only; one operator on each shift who tends to the ball mill, flotation machines, and other mill machinery; one filter man for each shift who also takes care of drying and sacking the concentrates; half of the time of a repairman on day shift, one assayer on day shift, and a superintendent, or a total of 9½ man-shifts per 24 hours. The assayer's time is only partly spent on mill work, the rest being devoted to assaying mine-ore samples and to duties in the store and warehouse.

MISCELLANEOUS

The mill is of frame construction, with concrete foundations and floors. It is heated by the dryer stove and two coal-burning heaters, one of which is also used to dry samples. Fire protection consists of soda-acid fire extinguishers and a fire hose connected to the high-pressure water line.

METALLURGICAL DATA AND COSTS

Table 4; a summary of concentrator costs for the same period is given in Table 5; and

Metallurgical data for the period February 1 to November 30, 1930, are presented in Table 6 shows costs in units of labor, power, and supplies.

Table 4.- Metallurgical data, February 1 to November 30, 1930

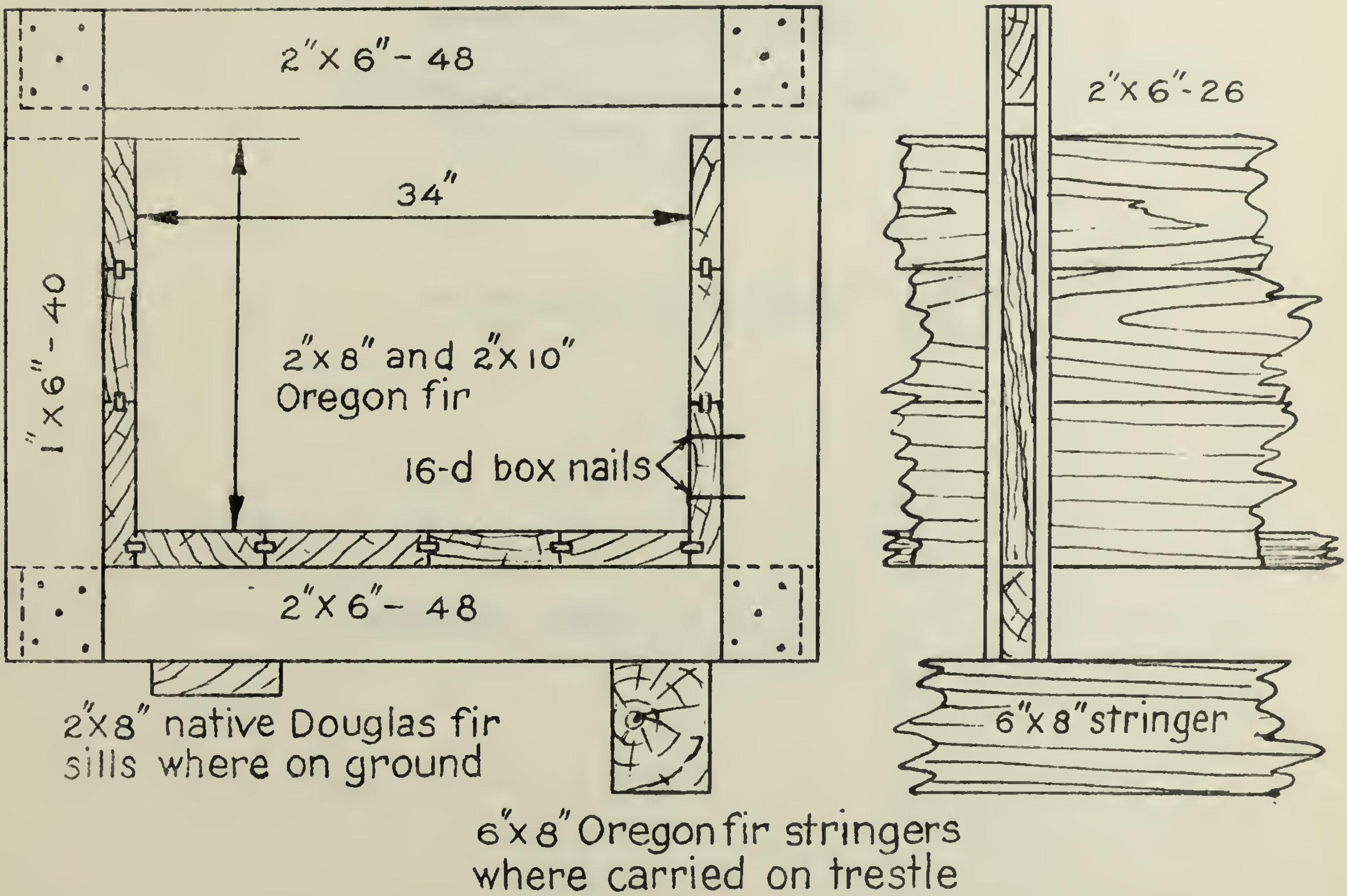
Ore treated, wet tons	11,239
Days operated	289
Hours operated per day	23.80
Average ore treated per day, tons	38.9
Concentrates produced, tons	673
Average concentrates produced per day, tons	2.33
Ratio of concentration, tons into 1	16.7
Feed, per cent of MoS_2	5.08
Concentrates, per cent of MoS_2	71.13
Tailings, per cent of MoS_2	0.68
Recovery of MoS_2, per cent	86.8
New water consumption per ton of ore treated, tons	2.5
Ball consumption per ton of ore, pounds	1.5
Liner consumption per ton of ore, pounds	0.934
Cyanide used per ton of ore treated, pounds	0.08
Flotation oil per ton of ore treated, pounds	1.00

Table 5.-Summary of concentrator costs, February 1 to November 30, 1930

	Cost per ton of ore treated
Breaking:	
Labor	$0.14
Concentrating:	
Labor, operating, including supervision	1.07
Supplies, operating	.90
Assaying:	
Labor	.12
Supplies	.06
Repairs:	
Labor	.20
Supplies	.05
Power	.15
Total	$2.69

Table 6.- Concentrator costs in units of labor, power, and supplies

	Tons per 8-hour man-shif	Man-shifts per ton
Labor:		
Breaking	39	0.0256
Milling, which includes grinding, classifying, and flotation	13	.0770
Dewatering and handling concentrates, including filtering, drying, and sacking	13	.0770
Tailings disposal	156	.0064
Assaying	39	.0256
Supervision	39	.0256
Miscellaneous	156	.0064
Average or total	4.11	0.2436
	Kilowatt-hours per ton	
Power:		
Breaking and elevating	1.1	
Grinding	20.0	
Classifying	2.6	
Flotation	13.3	
Dewatering and handling concentrates	5.5	
Tailings disposal	2.6	
Water supply	1.3	
Total	46.4	
Supply Consumption and Cost of Supplies:		
Reagents, pounds per ton:		
Cyanide	0.08 at \$0.24 per pound	= \$0.02 per ton of ore
Flotation oil	1.00 at \$0.73 per gallon	= \$0.09 per ton of ore
Miscellaneous:		
Balls, pounds per ton	1.5 at \$0.053 per pound	= \$0.08 per ton of ore
Ball mill liners, pounds per ton	0.934 at \$0.150 per pound	= \$0.14 per ton of ore
Sacks, number per ton	$1\frac{1}{2}$ at \$0.14	= \$0.21 per ton of ore
Coal for heating and drying, pounds per ton	22.4 at \$14.60 per ton	= \$0.18 per ton of ore
Other supplies		\$0.18 per ton of ore
Total operating supplies		\$0.90 per ton of ore



Note: Box is made of select common Oregon fir, surfaced on two sides, edges plowed 1/2 by 1/2 inch, in 16-foot lengths. Collars are of native pine and are spaced at 4-foot centers, every fourth collar having 4 by 6 inch top and bottom, and 4 by 6 inch fillers at the sides, to give nailing space at the box joints. Splines are of 7/16 by 15/16 inch white pine. The box is boarded over and heaped with earth, where possible, to prevent freezing. The grade is 1 1/2 feet per 1,000; average depth of water is 15 inches; velocity, 4 feet per second

Figure 4.- Flume for water-power plant

I. C. 6552

January, 1932

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

MINING LAWS OF EGYPT

BY

E. P. Youngman

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I.C. 6552
January, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

MINING LAWS OF EGYPT[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper presents one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of Egypt was prepared from the "Rules and Regulations as to Mining"[3] of the Egyptian Government and was checked against the answers made by the Egyptian Ministry of Foreign Affairs to a questionnaire of the United States Bureau of Mines, forwarded by W. M. Jardine, American minister at Cairo, and transmitted through the courtesy of the Department of State.

INTRODUCTION

Gold and precious stones were mined in Egypt in prehistoric times. This mining, carried on at intervals, extended over thousands of years, until about 1300 A.D. Then ensued a dormant period of several hundred years. In the meantime Europe made great progress in mining practice, metallurgy, and geology and sent out various expeditions during the nineteenth century to investigate ancient mines. But the exploratory work produced no material results, and not until the beginning of the present century did actual mining recommence. The reopening of old gold mines in Rhodesia aroused interest in London and Paris, and these cities provided the capital necessary for the investigation of the ancient mines in West Africa and Egypt. Speculation resulted in the division of most of the uncultivated area of Egypt and the Sudan into huge concessions, and a number of expeditions were made; but after an unprecedented activity in West Africa, reaction set in, and the flow of capital ceased. The concessionaires, mostly individuals with insufficient capital and with a misconception of the initial difficulties of communication, water supply, and labor, allowed

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1 - The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6552."

2 - Rare metals and nonmetals division, U. S. Bureau of Mines.

3 - Ministry of Finance, Egypt, Rules and Regulations as to Mining: Dept. of Mines and Quarries, Cairo, 1921, 94 pp.

many concessions to expire before they had been thoroughly prospected. This abandonment, coupled with misrepresentation concerning the gold mines, brought all mining in Egypt into disrepute.

In 1906, however, following the organization of the nucleus of a mining department in 1905, the present system of licensing and leasing superseded the old concession system, and from that time on gradual improvement has taken place in mining in Egypt.

In 1907 the Department of Mines was limited in expenditures to its earning capacity. To encourage an infant industry and to increase the revenue of the Government from this source without an excess in expenditures over earnings became the chief aims of the department. The first step taken was the standardization of agreements (licenses and leases). Their prompt issuance was insisted upon, and all attempts on the part of influential individuals or companies to obtain favors were resisted. The effect of this policy made itself evident in the gradual and steady rise of the revenue of the department, until in 1911 (after five years' work) it exceeded £ E.10,000 a year. Phosphate, manganese, and petroleum during subsequent years succeeded each other in economic development at the hands of foreign capitalists; and by 1928 the revenue to the Government from the mining industry had reached £ E. 83,500 a year, representing an increase in the earning capacity of the Mining Department in 21 years of £ E. 81,000 annually. During the 10-year period of 1918 to 1928, which may be considered the period of highest production, the excess of revenue over expenditures ranged from 50 to more than 100 per cent.

In 1910 the department took charge of quarries, which had been under no definite scheme of control, but which had been supervised by different Government agencies. Quarries in some instances had been granted on a life tenure, and none but the most primitive methods of stone extraction had been employed. Under a system of leasing, illicit quarrying has been checked, and the industry has been brought under such control that it is now yielding a much more adequate return to the State. The revenue from quarries in 1910 amounted to £ E. 1,803 and in 1928 (up to December 1) to £ E. 23,662.

In 1920 the department established an office for the investigation of petroleum; and in the summer of 1928, because of the receipt of numerous applications for permission to work the black sands on the northern coast of Egypt, it began to investigate the terms under which licenses and leases for that type of mining should be granted.

Although the publication of a mining law for Egypt is looked for within the next two years, according to an official Government report,[4] the new law is not to be a drastic revision of the existing code,[5] but rather a "crystallization of the present system of control into a definite statute." The

4 - Ministry of Finance, Egypt, Report of the Department of Mines and Quarries, 1928: Cairo, 1930, 44 pp.

5 - See footnote 3.

present system (that of granting general prospecting rights, with option to lease restricted areas) came into existence in 1906 upon the abolition of the old concession system, as has been indicated in a preceding paragraph.

RIGHTS OF FOREIGNERS

Foreigners are permitted to explore for minerals and to own and operate mines on equal terms with subjects of the Royal Egyptian Government. Foreign corporations need not incorporate under the laws of Egypt.

With respect to petroleum, the Egyptian Government reserves the right of preemption, or of preferential purchase, although all nationalities may engage in oil mining.[6]

Foreign labor also may be employed. In fact, in 1928 the monthly average was 125 foreigners and 3,034 Egyptians.

OWNERSHIP

The standard mineral mining lease and the petroleum mining lease both contain the following provision:

> Nothing contained in the lease shall be construed as vesting in the lessee the ownership of the land subject to the lease nor as conferring on the lessee any further or other rights over such land than those expressly granted to him nor as preventing the Government from dealing with such land in any manner it may think proper, which shall be consistent with the full exercise by the lessee of his rights under his lease.
>
> The lessee, upon the determination by effluxion of time or otherwise of the term of the lease, will deliver up peaceable possession of the land included in the lease to the Government or to any officer appointed by the Government to receive possession thereof.

The Government claims exclusive ownership of all mineral substances, whether they are found on public domain or on privately owned land. It will not sell its mineral territory but will grant long-term leases, in consideration of ground rent and royalty.[7] However, the question of the State's ownership of minerals on private lands has not been finally determined, and no precedent seems to have been established with respect to the rights of the owner of the land to the minerals thereunder, as ascertained from the reply of the Egyptian Ministry of Finance to the questionnaire of the U. S. Bureau of Mines.

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6 - Skinner, Robert P., American consul general, London, British Petroleum Policy: Consular Rept. 23703, Jan. 27, 1921; Bureau of Mines foreign file 2057.

7 - Van Wagenen, Theo. Francis, International Mining Law: McGraw-Hill Book Co. (Inc.), New York, 1918, p. 260-263.

MINING AUTHORITIES

The Department of Mines and Quarries, of the Ministry of Finance, is the mining authority in Egypt.

All applications for licenses or leases shall be made to the Controller, Department of Mines (Dawawin Post Office, Egypt), and all licenses and leases shall be issued by the Minister of Finance, upon the recommendation of the Controller.

APPLICATIONS

Each application for a prospecting license (or for an application reserve) shall be made in writing, the petition to specify the locality in which and the mineral for which the prospecting right is desired. The application shall be accompanied by a sketch map, showing as clearly as possible the location of the area or areas desired in relation to the sea coast or the Nile or some other distinctive feature of the landscape, together with the approximate distances and bearings therefrom. If the district has been officially surveyed, the sketch may be made on the plan or map thereof. Further procedure is given under the discussion of licenses and leases.

APPLICATION RESERVES

The registration of an "application reserve," as a convenience to a prospective applicant for a license, may be made (at the discretion of the Department of Mines and Quarries) for a maximum period of two calendar months.

Although the registration of an application reserve insures prior consideration to the holder thereof in his petition for a license, it is not a guarantee that a license will be issued to him.

PROSPECTING LICENSES

Mineral Prospecting Licenses

General conditions.—A mineral prospecting license, which shall cover only the mineral or minerals definitely specified in the agreement, is issued (for a consideration fee of £ E.[8] 25) subject to the existing rights of others. It gives no exclusive prospecting right to the holder, but it does insure to him prior consideration of his claim over opposing claims.

In order to obtain an exclusive prospecting right, the licensee,[9] within 60 days of the execution of his license, shall deliver to the Department

8 - Egyptian pound, which is worth approximately $5 in U. S. gold.

9 - The term "licensee" (whenever the context so admits) includes the licensee, his registered assignees, his or their legal representatives, and the agents, servants, or workmen of the licensee or of such assignees or representatives.

of Mines an official demarcation of the licensed area, on the form prescribed. He may then demand an official survey and a beaconing of the area by the Government. The demand shall be accompanied by a fee of £ E. 25 (increased by the department, if necessary), which is returned if a survey is not required.

If the Government gives its approval, the licensee has exclusive prospecting rights over the approved area and is entitled to a certificate of survey (if one has been made). If the Government refuses approval of the whole area applied for, the license shall absolutely "cease and determine," and the consideration fee and the survey fee shall be returned. If but a portion of the area is refused, the licensee is not entitled to the return of any moneys.

A licensee shall have the right at any time to apply for and receive a mining lease for all or any part of the area included in the license. As soon as a lease or leases of any portion or portions of a prospecting area have been granted, prospecting rights over the remainder of the area cease. (See, however, section of this paper entitled "Mining Lease Protection Agreements.")

Duration and renewal.--A license is issued for one year. It may be renewed for one-year periods, upon written application to the Department of Mines 15 days in advance of the expiration of the license or of a previous renewal, and upon the payment of a few of £ E. 25.

The Government is not bound to renew a license for more than one additional year after the expiration of the year during which the licensee shall (in the opinion of the Department of Mines) become entitled to demand a mining lease.

In order to obtain a first renewal of a license, the licensee must have started and continuously maintained bona fide geological and other prospecting operations; to obtain a second renewal, he must have started and maintained at least one working of sufficient capacity to reach and open the deposit and must have filed with the Department of Mines the required labor returns. (Operations are not considered to be continuous if suspended for longer than 30 days without the permission of the Government.)

Area.--The area included under a prospecting license shall be in the form of a rectangle and shall not exceed 1 mile in length or $\frac{1}{2}$ mile in width.

Disposal of minerals.--No minerals shall be removed under a prospecting license except such as may be necessary for the businesslike testing of the extent and richness of the deposit. After a lease has been granted, however, minerals produced in prospecting become the property of the lessee, subject to the royalty stipulated in the lease.

Transfer.--A license is transferable, but a transfer is not complete without a certificate from the Department of Mines to the effect that the stipulated sums have been spent in actual prospecting work. Thirty days are allowed for the registration of the instrument of transfer, and a fee of £ E. 5 is charged.

Petroleum Prospecting Licenses

General conditions.--Petroleum prospecting licenses differ but slightly from mineral prospecting licenses. A petroleum license is not transferable. The consideration fee is £ E. 100. No petroleum may be won, but natural gas raised in prospecting may be used for light and power. The other points of difference, which follow, are owing mainly to the nature of the mineral substance.

Conditions of renewal.--A petroleum prospecting license may not be renewed unless before the expiration of the original license the licensee shall have started and maintained bona fide geological and other prospecting operations. A second renewal will not be granted unless the licensee shall have started and maintained one boring of sufficient capacity, in the opinion of the Department of Mines, to reach, open, and take full advantage of the productive strata and shall have filed with the department the required labor returns. (Operations shall not be deemed to be continuous if suspended for more than 30 days without the written permission of the Government.)

Flows of water, gas, and petroleum.--A licensee shall take measures to control all flows of water, gas, or petroleum; and if the measures taken prove inadequate, the Department of Mines shall direct what measures the licensee shall adopt or shall itself take such measures at the licensee's expense (should he fail to comply with instructions).

Plans and bore sheets.--A licensee shall notify the Department of Mines of the position of any proposed borehole or shaft sunk by him and shall submit monthly to the department exact copies of bore sheets or shaft-sinking sheets, together with all data relating thereto. He should hold at the disposal of the department (should it so require) samples taken from such places in the boreholes or shafts as may be designated.

Supplemental reservos.--The registration of one or more "supplemental reserves" may be made, at the discretion of the Department of Mines, in the interest of the licensee of an adjoining area or areas.

A supplemental reserve may cover an area of the same size as that of a licensed area (2 by 2 kilometers) and may be registered only in conjunction with a specified license and only with respect to an adjacent area or areas, similar in geological structure to the area covered by the specified license.

MINING LEASES

Mineral Mining Leases

General conditions.--Normally a lease is granted as a right acquired under a prospecting license; but in special circumstances, when it is shown to the satisfaction of the Department of Mines that the mineral sought is present in sufficient quantities to justify a lease, a lease may be granted independently.

The holder of a prospecting license, at any time during the continuance of his license or of a renewal thereof, may apply for and receive a mining lease for all or any part of his prospecting area. The size of the leased area is designated in the lease; the form thereof shall be rectangular, and the length shall not exceed twice the width. It shall cover only such mineral as is mentioned specifically therein.

The granting of a lease or leases for the whole or any part or parts of a prospective area renders the prospecting license void, and the license is surrendered to the Government. (See, however, the section of this paper entitled "Mining Lease Protection Agreements.")

A lease grants to the lessee[10] the exclusive right to "search, dig, mine, get, make merchantable, and carry away" the prescribed mineral or minerals in and "vertically under any or every part of the piece of land described in the schedule and shown on the plan annexed to the lease," together with "all such powers of sinking shafts, driving headings, making watercourses or wells, laying down tramways, making roads, and erecting or constructing machinery, buildings, or other works . . . as may be necessary or proper for taking full advantage of the lease."

Duration and renewal.--A lease is granted for 30 years and is subject to a renewal for 10 years, if at the expiration of the lease the Government is satisfied that the lessee has duly observed all the terms thereof.

Work required.--A lessee, within four months of the issuance of the lease shall commence and thereafter carry on continuous and businesslike operations. (Work is not deemed to be continuous if interrupted for more than 30 days without the consent of the Department of Mines.)

Records.--A lessee shall keep correct plans of all mine workings and accurate records of all work done and of the amount and value of all metals and minerals extracted; he shall likewise construct and maintain all such works and machinery as may be reasonably necessary to show accurately the amount of all substances raised and saved.

10 - The term "lessee" (whenever the context so admits) includes the lessee, his registered assignees, his or their legal representatives, and the agents, servants, and workmen of the lessee or such assignees or representatives.

Transfer.--A mining lease may be transferred, but only with the authorization of the Minister of Finance. A period of 30 days is allowed for the registration of an instrument of transfer, and a fee of £ E. 5 is imposed therefor.

Petroleum Mining Leases

General conditions.--Every holder of a petroleum prospecting license shall be entitled, at any time after approval of his prospecting area and during the continuance or the renewal of the license, to apply for a mining lease or leases, subject to the following conditions:

1. Each lease shall be for some portion of the prospecting area.

2. The total area comprised in the lease or leases shall not exceed 100 hectares.

3. Each separate area comprised in any one lease shall be in the form of a rectangle, no side of which shall be less than 500 meters long, unless the Department of Mines shall have authorized in writing some other form or dimensions.

4. No lease shall be granted of any area unless it has been proved (to the satisfaction of the Department of Mines) that it contains at least one productive well.

5. A licensee shall beacon each area with respect to which a lease is demanded, in accordance with the regulations in force.

6. A lease shall be in the form indorsed on the license.

The granting of a lease or leases for any part or parts of a prospecting area renders void the prospecting license for the remainder of the area. The license is surrendered in exchange for the lease. Although, in principle, a mining lease may be granted only as a right acquired under a prospecting license, nevertheless after the reversion to the Government of the remainder of a licensed area upon the grant of a lease, the lessee will be given the prior right to a license or a lease over the whole or the remainder of the original licensed area, subject to a royalty not to exceed 25 per cent.

A petroleum mining lease grants to the lessee the exclusive right to search for, dig, mine, get, and carry away all petroleum and petroleum gas lying in and under any part of the piece of land described in the schedule and shown on the plan annexed to the lease, together with "all such powers of sinking shafts and wells, laying down, using, working, and removing railways, tramways, or pipe lines, obtaining, conveying, and using water and gas, making roads, and erecting or constructing and removing machinery, buildings, or other works upon such land as may be necessary or proper for the raising and storage of petroleum and petroleum gas and for the transport thereof-

within the area of such land." The Government will, if required, grant to the lessee (by a separate agreement or agreements) all such further rights as may be necessary or proper to enable him to provide for the refining of the petroleum and petroleum gas raised and to take full advantage of the lease.

Duration and renewal.--A lease is granted for 30 years, and it may be renewed for 15 years upon the following conditions:

1. If at the expiration of the lease, the lessee is able to prove that he has observed all the terms of the lease and that it has not been liable to be declared void under any of the provisions thereof.

2. If the lessee shall have given to the Minister of Finance a 6-month notice in writing of his intention to renew.

3. That a renewal shall be subject to the same rents, conditions, and provisions as was the lease, except that on any renewal the Government may impose such rate or royalty (not exceeding 25 per cent ad valorem) as it may think proper.

Work and workings required.--The lessee of a petroleum right shall commence work within four months of the date of his lease and shall thereafter continue businesslike operations. (Work shall not be considered to be continuous if suspended for more than 30 days without the written permission of the Department of Mines.) However, temporary suspensions are provided for, under special rulings, when market conditions are prohibitive of profitable operations.

A lessee shall maintain in continuous operations at least two boring rigs of sufficient capacity to take advantage of the lease for a period of 20 years or until there shall be in each 10 hectares of the leased area at least five boreholes, either actually productive or potentially so from the geological nature of the field.

Records.--A lessee shall keep correct plans of all workings within the leased land, an accurate record of every borehole or shaft sunk, and all proper books of account and all other books necessary to show the amount and the value of all petroleum and petroleum gas raised. He shall construct and maintain all such works and machinery as may be necessary (in the opinion of the department) to show accurately the amount of all substances raised and saved.

Transfer.--A petroleum mining lessee shall not sublet in whole or in part any of the rights granted by his lease or assign any of such rights without the written consent of the Minister of Finance, which consent will be given on the following conditions:

1. The terms and conditions of the lease shall have been duly observed and the rents and royalties duly paid.

2. The area with respect to which such rights are to be assigned shall not be less than 10 hectares in extent, and the length thereof shall not exceed four times the breadth, unless the Department of Mines shall have authorized in writing some other form or dimensions.

3. Proof shall be established (to the satisfaction of the Department of Mines) that the area to be assigned contains at least one productive well or that some other evidence of the presence of petroleum exists.

4. The proposed assignee shall produce proof (satisfactory to the Department of Mines) that he is financially capable of undertaking the necessary work. For this purpose, an available working capital of at least £ E. 250 per hectare (with a minimum of £ E. 10,000) shall be required.

5. Every instrument of assignment shall contain sufficient provisions for binding the assignee to observe all the covenants and conditions contained in the lease, with any necessary modifications; and for this purpose a draft of the proposed assignment shall be submitted to the Department of Mines for approval.

A period of 30 days is allowed for the registration of an instrument of transfer, and a fee of £ E. 5 is charged therefor.

Caution money.--A lessee shall deposit in the treasury of the Department of Mines the sum of £ E. 100, as "caution money," which shall be refunded to him at the expiration of the lease, provided that all the terms of the lease and all the regulations have been complied with. This caution money shall bear no interest, and it shall be liable to confiscation by the Government in case of any infraction or default on the part of the lessee.

Government's preferential right of purchase of products.--The Government shall have a preferential right, on two months' notice in writing, to purchase from the lessee so much of the crude petroleum as it may require, provided that the amount the Government shall be entitled to purchase in any one year shall not exceed 20 per cent of the amount produced on the land during the preceding year. The price to be paid by the Government shall be 10 per cent below the latest price ascertained for the payment of royalties (see section of this paper entitled "Rents and Royalties") for so much of the petroleum purchased as shall not exceed 10 per cent of the output of the previous year and shall be such latest price for the remainder of the purchase.

MINING LEASE PROTECTION AGREEMENTS

A mining lease protection agreement may be obtained for the balance of a prospecting-license area not taken under a mining lease. The area shall not be more than double that of the lease protected, and its length shall

not exceed twice its width. No protection agreement shall be granted to protect a lease of less than 60 acres for sedimentary deposits or of less than 25 acres for lode deposits.

The terms of a mining lease protection agreement are practically the same as those of a prospecting license. (See pages 28 to 35 of Rules and Regulations as to Mining.[11])

TERMINATION OF LICENSES

Surrender

A licensee of either a mineral prospecting or a petroleum prospecting right may at any time, by written notice to the Minister of Finance, surrender his license. Such surrender shall be without any prejudice to any claim against the licensee that shall have accrued to the Government prior to the giving of the notice, and particularly without prejudice to the right of the Government to retain any sum already paid under the license.

Forfeiture

The Government may cause the forfeiture of a license (1) if the licensee shall fail to conform to any rule or regulation in force and shall fail to remedy the breach within one month of the notice from the Department of Mines or (2) if the licensee shall be declared bankrupt or insolvent by a competent tribunal or, being a company, shall go into liquidation or be dissolved.

The voidance of a mineral prospecting license shall be deemed to have been legally served by its publication in the Official Journal of the Egyptian Government. A forfeiture shall vest in the Government all the buildings, plant, and other property of the licensee that may be upon the land, without the payment by the Government of any compensation, provided that if the license shall be forfeited for any cause other than the nonpayment of fees the licensee shall be entitled within two months from the date of the notice to take away all movable plant.

The voidance of a petroleum prospecting license shall be made by written notice from the Minister of Finance, and thereupon all the rights of the licensee shall absolutely cease, without prejudice to any claim of the Government against the licensee.

TERMINATION OF LEASES

Surrender

A lessee of a mineral mining right may at any time terminate his lease by giving six months' notice in writing. All buildings, plant, and other property left upon the land at the time of the termination of the lease shall become the absolute property of the Government, without any payment or compensation to the lessee therefor. The automatic termination of a lease by

11 - See footnote 3.

the expiration of the period for which it was granted entitles the lessee to three months in which to remove buildings, plant, and other property.

The provisions for the surrender of a petroleum mining lease are practically the same as those for the surrender of a mineral lease--the main difference being that the holder of a petroleum lease is entitled to a proportionate abatement of the rent payable. (See section of this paper entitled "Rents and Royalties.")

Forfeiture

Either a mineral mining lease or a petroleum mining lease may be declared void, without prejudice to any claim against the lessee accruing to the Government under the lease (1) if the lessee shall have committed any breach of regulations or shall have failed to pay rent, royalty, or any other sum due and shall not have remedied the breach or paid the sum within two months from the receipt of notice of his delinquency, (2) if the lessee shall purport to assign the benefit of his lease (or in the case of a petroleum lease shall seek to assign or sublet his benefit) without the written consent of the Minister of Finance, or (3) if the lessee shall be declared bankrupt or involvent or, being a company, shall go into liquidation or be dissolved.

If a lease is voided for any other cause than the nonpayment of rent or royalties, the lessee shall be entitled at any time within two months from the date of the notice of forfeiture to take away all movable plant that was on the land at the time of the notice.

Publication in the Official Journal of the Egyptian Government of the decision to void a lease shall be considered sufficient notice.

RENTS AND ROYALTIES

Mineral Mining Leases

Rent.--The amount of ground rent due the Government is designated in each lease at so many Egyptian pounds per acre or fraction thereof; it is payable in advance on the first day of January.

Royalty.--The amount of royalty, too, is specified in each lease, as a certain percentage of the value of the mineral at the pit's mouth; it is payable within two months of the first day of January in each year with respect to the minerals extracted during the previous 12 months.

Determination of the value shall be based upon the market price of the mineral upon the London market, as quoted in reputable trade journals, less the average cost of transport to London; or (at the discretion of the Department of Mines) it shall be based upon the actual price obtained by the lessee. (The lessee shall furnish, on demand, such data, certified by a chartered accountant, as the Department of Mines may require for ascertaining the amount of mineral extracted or the price obtained therefor.)

If the royalty calculated in any year shall be equal to or less than the rent due, no royalty shall be required for that year; if the royalty calculated shall exceed the rent, only the excess of the royalty over the rent shall be demanded.

Petroleum Mining Leases

Rent.--A yearly rent of £ E. $2\frac{1}{2}$ per hectare of land or fraction thereof is payable in advance, on the first day of January.

Royalties.--A royalty of $12\frac{1}{2}$ per cent of the petroleum and petroleum gas raised and saved during the previous six months is payable half-yearly, on the first day of January and the first day of July. The cost of the delivery of the products rendered as royalty to such place as the Government may direct is borne by the Government. The lessee shall store such products free of expense to the Government for two months and thereafter at the current storage rent of the oil field, if there is any, and at a rate agreed upon, if there is no established rent.

If the Government shall so require, the value of the products rather than the products shall be paid. The value shall be calculated on the average market price during the period for which the royalty is due. If no market price is ascertainable, the lessee shall pay the price fixed by the Department of Mines after consultation with the lessee or his representative.

In computing the amount of royalty due for any period, the amount of rent payable for that period shall be deducted, provided that, if during such period the lessee shall not have raised and saved a quantity of petroleum and petroleum gas the royalty upon which is equal to the rent due, the lessee shall not be entitled to deduct the amount of such deficiency from the royalty for any subsequent or other period.

MISCELLANEOUS OBLIGATIONS

Regulations with respect to demarcation of areas, use and storage of explosives, standard forms for certificates, permits, and reports, and similar matters are covered in pages 57 to 94 of Rules and Regulations as to Mining.[12] The Department of Mines lays down for the guidance of operators general rules designed to protect the health and safety of the workers; for the rest the manager of each company is responsible to the department.

Among the miscellaneous obligations actually embodied in the licenses and leases themselves are the following:

Expert management.--A technically competent manager must be employed "on the spot" at all times, and notice of his appointment must be given to the Department of Mines.

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12 - See footnote 3.

Returns.--Returns with respect to the labor employed and the minerals produced must be made monthly to the Government.

Policing expenditures.--Unless a special or general tax shall have been levied for the maintenance of order and the carrying out of sanitary regulations at the place, or in the neighborhood, of prospecting or mining operations, a licensee or a lessee shall pay an equitable proportion of the Government's policing expenditures.

Damages.--A licensee or a lessee shall be entirely and solely responsible toward all persons for any damage caused by his operations, and he shall keep the Government indemnified against all actions, proceedings, claims, and demands with respect thereto.

Antiquities.--All objects of antiquity discovered during prospecting and mining operations must be turned over to the Government and must be safeguarded until such time as the Government shall act.

TAXES, DUES, AND EXCISE

A lessee shall promptly and regularly pay all taxes, dues, and excise already or hereafter lawfully established, and nothing in the lease shall be interpreted as limiting the Government in its right to establish or impose any later tax, due, or excise within its competence or as exempting a lessee from the obligation to pay it.

Excise duty on petroleum products of Egyptian origin was decreed March 30, 1921, the amount to be fixed monthly, in advance.[13]

QUARRIES[14]

Quarry leases are issued for periods of 1, 5, or 10 years, and the rents vary with the quality of and the demand for the product and with the accessibility of the quarry. The area now leased in the case of stone quarries is normally 5,000 square meters, whereas for sand, gravel, and gypsum the area is variable in extent.

Permits for the erection and maintenance of decauville railways (aerial ropeways) and limekilns are issued in connection with quarry leases.

In order to facilitate administration and to adopt regulations suited to varying local conditions, the quarries of Egypt are now divided into four principal groups; Upper Egypt, Cairo, Lower Egypt, and Mex. The islands in the bed of the Nile River also are divided into groups for administrative purposes, and they are rented for the extraction of sand.

13 - Maynard, Lester, American consul, Alexandria, Excise Duty on Petroleum Products: Consular Rept. 74130, Oct. 13, 1922; Bureau of Mines foreign file 4907.
14 - See footnote 4.

A certain number of quarries, reserved for the use of the State, are allotted to Government contractors, who pay a royalty on the stone extracted, or they are developed by the Government departments. This system assures a supply of the best available stone and protection against price raising.

PRESENT STATUS OF THE MINING INDUSTRY

That Egypt has now a well-established and steadily growing mining industry is indicated by the fact that the Government has decided to publish an annual report covering statistics and general information with respect to that industry. Heretofore reports were published only at 5-year intervals. The first of the yearly issues is that for the year 1928, published in 1930,[15] which deals chiefly with phosphates, manganese, and petroleum, the three mineral substances that, in connection with the products of the quarries, furnish the major mineral production at present, and the development of which during the past 20-odd years has enabled Egypt to take a definite, if minor, place among the world's producers of minerals. (This first annual gives, in addition to the present status of each mineral, historical and descriptive facts that will not be found in subsequent numbers.) The report covers other minerals also, which either have been produced in the past, are being produced in small quantities now, or are covered by prospecting licenses. These are gold, peridots (gem variety of olivine), emeralds, nickel, molybdenite, lead and zinc, titaniferous iron sands, crude carbonate of soda, nitrate shale, pigments talc, and clays.

A comparative table of the production of the chief minerals for 1928, 1929, and 1930 follows, as well as a table showing mineral exports for 1930.

Comparative table of the principal mineral production in Egypt, 1928, 1929, and 1930[1]
(Metric tons)

Minerals	1928	1929	1930
Petroleum - - - - -	268,323	272,114	285,088
Phosphate rock - - -	200,563	215,311	313,478
Manganese ore - - -	137,502	191,477	128,211
Ocher - - - - - - -	944	1,017	1,184

1 Russell, H. Earle, American consul, Cairo, Mineral Production (Review of Commerce and Industries for the Year 1930): Consular Rept. 20450, Apr. 6, 1931, pp. 25-29; Bureau of Mines foreign file 4913.

15 - See footnote 4.

Mineral exports from Egypt in 1930[1]

Commodity	Quantity, metric tons	Value, Egyptian pounds[2]
Metallic ores - - - - - - - - (manganese, etc.)	98,863	124,207
Caustic soda - - - - - - - -	140,477	2,105
Common salt - - - - - - - - -	154,852	23,371
Phosphates and super-phosphates - - - - - - - -	8,433	5,987
Phosphate, lime - - - - - - -	314,170	252,152
Petroleum products:		
Benzene - - - - - - - - -	53,174	380,111
Kerosene - - - - - - - - -	423,826	1,660
Solar and gas oils - - -	2,460,949	6,120
Fuel, Diesel oils - - - -	31,169	59,371
Coke, pitch, asphalt - - (bitumen) of petroleum	58,067	278,496

1 Russell, H. Earle, Work cited.
2 The exchange value of the Egyptian pound in U. S. currency in 1929 was $4.98

Unskilled surface labor is plentiful at the rate of about 10 P. T.[16] a day, higher or lower according to local conditions. Underground rates are somewhat higher, and wages paid to miners under contract average 15 P. T. a day. Skilled artizans receive from 15 P. T. to 50 P. T., and clerical workers earn a minimum of 20 P. T. a day.

16 - Turkish piaster, which is equal to 1/100 of the Egyptian pound.

I. C. 6953
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INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

HAZARDS OF COMPRESSED-AIR JETS FOR VENTILATING GASSY MINES[1]

By R. D. Currie[2] and L. L. Naus[3]

INTRODUCTION

Hazards in connection with the use of compressed-air jets for ventilating gassy mines are given little consideration by most mine officials largely because compressed air is considered the safest means of transmitting power for operating various types of mining machinery, and the inherent dangers in its use for this purpose are virtually negligible.

The installation of compressed-air lines in coal mines leads to the utilization of compressed air for ventilating working places, with the result that, although the primary ventilation systems may be adequate to meet existing needs, proper conduction of normal ventilation has been neglected, so the working faces receive only a small amount of the total air circulated through the mine.

Reliance in part on compressed air or other auxiliary means for ventilation is always attended by danger that gas will accumulate, due to intermittent operation, and the tendency is to operate auxiliary schemes only during the working shift (or possibly only part of the shift) and to dispense with them during off-shifts or at times during the working shift when gas may accumulate because of lack of ventilation.

When the men come on shift the auxiliary ventilating device often may be started without a preliminary examination of the working place, or the examination may be made sometime before the oncoming shift arrives, the gas being allowed to accumulate in the meantime. This may result in uncontrolled movement of a body of gas if proper precautions are not taken. Several mine explosions have been caused by the movement of a known body of gas by compressed air into the haulageway, where it was ignited by an arc from a passing locomotive. This danger is more pronounced where supervision is lax.

The high velocity of compressed air escaping from jets, open valves, or open-end supply lines may cause a decided churning action, which serves to

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1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgement is used: "Reprinted from U. S. Bureau of Mines Information Circular 6953."
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mix more intimately any liberated gas or dust and greatly increases the danger of ignition from these sources. It also tends to maintain dust in continual suspension, which cannot be other than detrimental to the workman. High-velocity compressed air issuing from jets may create channels through gas accumulations in such manner as to remove or dilute only a part of the accumulations, leaving much of them to be ignited later.

Experience has shown that the common methods of using compressed air for ventilating mines are extremely hazardous, in addition to which it has generally been accepted that they are inefficient and ineffectual. The following accidents, in which compressed air used for ventilation was directly responsible for accumulation of gas, are representative of scores of such accidents reported to the Bureau of Mines:

1. A gas explosion occurred at an anthracite mine on January 10, 1936, as the direct result of using compressed air for ventilation.

A 3/4-inch pipe, swaged down to a 1/16-inch opening and supplying compressed air at approximately 55 pounds pressure, was being used to ventilate a slant heading between two breasts being driven on a 45° pitch. The slant was about 300 feet from the gangway.

The board manway brattice was loosely constructed and too far back from the face of the breast to supply ventilation in the proper manner, and the compressed air jet was about 15 feet back from the face.

A safety lamp carried by the miner had been extinguished by an explosive mixture of methane and was being held at the nozzle of the jet to clean out the lamp and cool it; at the same time the igniter was used to re-light the lamp. An explosion occurred that resulted in the death of the two men in the place.

This lamp was tested in a McCaa lamp-testing box immediately after the explosion; no explosion could be produced with it in the tests. The conditions prevailing at the time of the explosion were duplicated as nearly as possible in tests conducted by the Bureau of Mines, and it was found that a properly assembled flame safety lamp of permissible type will ignite explosive gas under such conditions.

This practice of cleaning out and cooling flame safety lamps with the compressed-air jet is decidedly hazardous although quite common in some sections of the anthracite field.

2. On April 19, 1909, upon detecting gas at a working face in a coal mine in Ohio, a fire boss opened a compressed-air valve to ventilate the place and left. No danger board was erected, and a man wearing an open light entered the place and ignited the explosive mixture.

3. On June 28, 1927, nine men were badly burned at a mine in Alabama by an ignition of gas caused by an arc in the 110-volt light circuit about 12 feet back from the upper end of a slope that was being driven.

Methane accumulated after the rock-drill compressor supplying the only means of ventilation had been shut off, as was customary, for 4 hours just before the men came on shift.

4. On November 14, 1927, a raise driven off the engine room to the coal seam in a bituminous coal mine in Washington was giving off gas at the face. A compressed-air hose was used to supply ventilation and to prevent the accumulation of the gas. While making his rounds, the fire boss found gas filling the raise and extending nearly to the haulageway. The compressor was shut down and the place fenced off by the fire boss, who then waited the arrival of the mine foreman to receive further instructions. The fire boss was instructed to blow the gas out by using compressed air. He had started to do this when a passing locomotive ignited the gas, with the result that four men were burned, one of whom died later.

5. On April 12, 1930, 17 men were killed by a gas explosion at the above-mentioned mine. This explosion happened in a chute that was ventilated by compressed air and a ventube blower, and at least two of a series of holes were overcharged with permissible explosives. The shots were fired in multiple with a 500-volt trolley wire.

Coal dust no doubt was thrown in suspension; gas probably was present or was released by the blast of the first shot, and the flame, presumably from the detonation of the overcharged second shot, ignited the dust, gas, or both.

6. An explosion that occurred at a gassy bituminous mine in Utah on March 8, 1930, burned seven men, with the result that five of them died and the other two were injured seriously.

The fan was not running at the time of the explosion. Under normal working conditions the fan was run only intermittently and natural ventilation was supposed to ventilate the mine when the fan was not in operation.

Compressed-air jets were used to ventilate the working places and no line brattice was carried to the face. It is probable that the explosion was caused by ignition of an accumulation of gas by a spark or arc from a nonpermissible mining machine. As water was used on the cutter bar, it is not likely that coal dust played more than a minor part in this explosion.

7. On June 21, 1933, an explosion occurred at a bituminous coal mine in Oklahoma which resulted in the death of the mine foreman and one of a group of four rescuers.

The explosion was the result of use by the mine foreman of a compressed-air hose to clear gas from a vertical cut before blasting. The mixture of gas with coal dust, raised in suspension while the kerf was being cleaned, probably was ignited when the foreman attempted to light the shot with a match.

8. On January 12, 1935, an accident occurred in the sand blasting department of a California manufacturing company, which caused the death by carbon monoxide of one of three sand-blast operators, who were furnished ventilation to sand-blast masks from a 5,000-cubic-foot compressor through a 1/4-inch heavy hose. One operator detected a strange odor and left his compartment to check with the other two. He found one of them absent from his compartment and the other dead from carbon monoxide poisoning.

Due to a broken exhaust valve, the compressor became unduly heated and set fire to oil and carbon, and the carbon monoxide liberated was carried through the hose to the masks.

These accidents, in which the use of compressed air for ventilation was directly involved, indicate that it is not a safe means of ventilation, at least in coal mines, which give off explosive gas, and at best is a decidedly costly method per unit of volume of air furnished or moved.

Another hazard in connection with the use of compressed-air jets in gassy and dusty mines is that of possible static sparks. Although this hazard has not been recognized in this country, it has been given considerable thought and study in Europe, as indicated by the following information.

In Belgium and France firedamp explosions have occurred during the dispersal of gas accumulations by compressed air, the ignitions having been credited to electrostatic charges generated at the nozzle by the issuing air[4].

Tests made by passing clean or dust-laden compressed air through an iron tube at different velocities (20 to 50 meters per second) and pressures (4 to 8 atmospheres) revealed that the dust particles and the nozzle with supplementary fittings acquired opposite and equal electrostatic charges.

Due to friction along the walls of the tube, compressed air alone gave potentials of only 20 volts, while dust-laden air gave potentials of several thousand volts. When the air is humid, a mist is formed in front of the nozzle and electrostatic charges are not likely to be produced.

Conditions most favorable to the generation of these charges were dry nonagglomerating dust in a concentration of about 200 grams per cubic meter; dry compressed air; air velocities over 20 meters per second; dry, warm, low-ionizing air surrounding the charged particles; and a satisfactory insulation.

The intensity of spark required to ignite gas was studied on a gas mixture containing 7 to 8 percent of methane; in dry warm weather ignition was obtained in each test within a few seconds after turning on the compressed air. In damp weather an ignition was much more difficult to obtain, owing to the amount of moisture in the atmosphere and its high state of ionization.

4/ Iron and Coal Trades Review: Electrostatic Charges Generated by Compressed Air, November 24, 1933, p. 797.

Although the tests showed that the sparks will cause ignition, it is concluded that the requisite conditions for an ignition occur rarely in underground workings. In cold weather, electrostatic charges may be produced underground at compressed-air nozzles. It also may be assumed that the air in dry, hot workings will have a low conductivity, and thus favor the generation of electric sparks.

As the compressed-air mains are usually sufficiently grounded, no high potentials can be produced by them. The danger lies in well-insulated metallic parts.

The pipes used to clean boreholes, especially if joined to the supply by a rubber pipe or hose, are a source of serious electrostatic charges and hence may be the possible cause of ignition. Grounding the metal pipes will obviate all danger.

Experiments have shown that enough electricity can be generated by the friction of a small pulley and belt (such as booster fan drives) to ignite natural gas readily.

Static electrical charges of rather high potential are sometimes generated by leaks in steam and air lines. These charges are equalized by sparks jumping from the charged cloud of steam or air, which may be of sufficient intensity to ignite a mixture of methane and air. It is not definitely known whether such a spark could ignite coal dust.

The following table gives the discharge of air from small orifices; these figures have been used to calculate the flow from jets under conditions observed in the anthracite region of Pennsylvania.

Gage pressure, pounds	Discharge, cu. ft. of free air per min.		
	Diameter of orifice, inches[5/]		
	1/16	1/8	1/2
40....................	3.07	12.3	196
45....................	3.36	13.4	215
50....................	3.64	14.5	232
60....................	4.20	16.8	268
70....................	4.76	19.0	304
80....................	5.32	21.2	340
90....................	5.87	23.5	376
100....................	6.45	25.8	412

Under average conditions of pitch, height, and width, the minimum quantity of air necessary to ventilate properly a gassy, two-man, working place is 400 cubic feet per minute. A compressor with a capacity of 2,000 cubic feet of free air per minute would be capable of supplying compressed air to only five such working places.

5/ Peele, Robert, Mining Engineers' Handbook, 1927, p. 1090.

The proper ventilation of a gassy mine, or portion of such a mine, solely by compressed air is not practicable because of the prohibitive cost. At 3 cents per 1,000 cubic feet for compressed air, which is probably a fair average, the cost of ventilating a gassy mine of five two-man working places for 8 hours would be $28.80. To ventilate a moderate sized gassy mine of 100 two-man working places by compressed air would cost a minimum of $576.00 per 8-hour day.

Based on the preceding table, the cost of using compressed-air jets is as follows:

Gage pressure, pounds	Cost per hour, cents		
	Diameter of orifice, inches		
	1/16	1/8	1/2
40.................	0.55	2.21	35.28
45.................	.61	2.41	37.70
50.................	.66	2.61	41.76
60.................	.76	3.02	48.24
70.................	.86	3.42	54.72
80.................	.98	3.82	61.20
90.................	1.06	4.23	67.68
100.................	1.16	4.65	74.16

During 1932, 43,826,027 tons of coal were produced in the Pennsylvania anthracite region by 94,210 underground employees working an average of 157.5 days per man. The average production per underground employee was 2.9 tons per day.6/

A gassy mine operating five two-man working places would produce 29 tons of coal per day; if this mine were ventilated by compressed air at a cost of $28.80 per day, the cost of ventilation per ton of coal produced would be 99.3 cents. On the other hand, the cost of ventilation by a small unit fan and galvanized iron tubing is 0.03 cents per 1,000 cubic feet7/, or 0.993 cents per ton, based on the above fitures of production. Hence, this unit cost of supplying compressed air is about 100 times as great as the cost of supplying the same quantity of air by fan ventilation.

Although it is apparent that no company would attempt to use compressed air to ventilate a mine entirely, there are mines in which compressed-air jets, open-end compressed air lines, and other subterfuges for adequate ventilation are used or are permitted to be used. It is believed that such practices are the result of failure of mine officials to conduct the ordinarily sufficient, normal ventilation to the working places by properly maintained brattice or to follow a seriously considered ventilation program.

6/ Adams, W. W., and Geyer, L. E., Coal-Mine Accidents in the United States, 1932: Bull. 380, Bureau of Mines, 1934, 87 pp.
7/ Harrington, D., Metal-Mine Ventilation and Its Relation to Safety and Efficiency in Mining Operations: Report of Investigations 2133, Bureau of Mines, 1920, 8 pp.

A recent study has revealed that use of reducer nozzles or jets at the end of supply lines is common practice in anthracite mines. Jets of 1/2-inch opening were found on gangways where the compressed air was used for several hours after blasting; jets of 1/8-inch opening were used in chutes and breasts and were allowed to remain in use at all times (in some instances, even on idle days); jets were sometimes found inserted in 5- by 6-inch wooden tubing (inside measurement) about 6 feet long, producing an injector effect; usually, however, they were connected to the supply line without the tubing and too far back from the face to be of benefit.

Although compressed-air jets in a box or tube created an injector effect and circulated a considerably larger quantity of air, in virtually every instance the result was recirculation of the air in the working place.

One installation consisted of three 1/16-inch jets spaced about 100 feet apart along the supply line in a pillar hole that had been driven about 430 feet. The pressure on the line was about 40 pounds, so that the maximum free air delivered from the three jets could not exceed 10 cubic feet per minute.

At another place on a gangway, a 1/2-inch jet inserted in the end of metal tubing 24-inches in diameter and 100 feet long extended from the gangway door about 135 feet outby the face. The quantity of air delivered at the inby end of the tubing was 1,866 cubic feet per minute, of which 376 cubic feet was delivered from the compressed-air line, according to the table for a 1/2-inch jet at 90 pounds pressure.

A similar installation consisted of a 1/8-inch jet inserted in the end of metal tubing 8 inches in diameter leading from the gangway to the working place through a rock hole. In this instance the 8-inch metal tubing had an extension of 5- by 6-inch wooden tubing. The quantity of air delivered in this instance was 466 cubic feet per minute, of which 24 cubic feet was supplied by the jet, according to the table for a 1/8-inch jet, at a pressure of 90 pounds.

Other methods for ventilating by compressed air consisted of open or partly open valves and open-end supply lines, which sometimes were inserted in one end of a 6-inch length of wooden tubing.

As commonly used in mines, ventilation by compressed-air jets, open valves, and open-end supply lines is neither efficient nor effective; because of the high velocity of the escaping air and the turbulent action created, it requires at least 6 volumes of compressed air to remove 1 volume of smoke such as is created by blasting. On the other hand, 1 volume of fresh air conducted to a working place by means of a properly constructed brattice so as to maintain a velocity of about 25 feet per minute, will remove easily 1 volume of smoke.

An adequate and dependable ventilating system precludes the necessity of secondary ventilation schemes such as compressed-air jets, open valves, and open-end compressed-air supply lines. In general, it will be found that wherever compressed-air lines are installed there is a tendency on the part of the men to use the compressed air in an effort to clear away the smoke from blasts and for ventilating purposes, simply because the ordinarily sufficient, normal ventilating current has not been maintained properly. When compressed-air jets, booster fans, and ventube blowers are used or are permitted to be used in normal mining operations, it is essentially a confession on the part of the mine officials that their primary ventilating system is inadequate or undependable.

Some companies have taken definite steps to eliminate the costly and wasteful use for ventilation of compressed air for power tools. A reducer (shown in figure 1) has been installed by one company at all junction lines where compressed air is used for drilling; this limits the amount of air allowed to escape should the valve be left open after the jack-hammer is disconnected.

The valve shown in figure 2 apparently is a better, although more costly, means of eliminating leakage. When this valve is installed in the compressed-air line, the only leakage possible is through the very small bypass opening in the valve, which allows a back pressure to build up against the valve until the pressure on each side of the valve stem is equalized when a jackhammer is connected to the end of the line. The adjusting spring is set so that the valve remains open while the jackhammer is connected. As soon as the connection is broken, the pressure on the supply side of the valve causes the stem to seat and shut off the flow of air, except the small quantity that flows through the bypass opening. The valves are adjusted and sealed in the shop so that they cannot be disturbed; in case they fail to function properly they are sent back to the shop for repair.

There are places and conditions that may justify the use of secondary ventilating aids, as in prospecting, starting new places, cleaning sumps, driving rock tunnels, or sinking or raising shafts.

Under such conditions comparatively safe and efficient methods of using compressed-air power for driving ventilating machinery are compressed-air-driven fans, blowers, and injectors.

These devices, however, should be placed so that they do not recirculate the mine air and will be in an air stream of at least four times their capacity, or are bratticed off in such a manner as to preclude the possibility of recirculation.

These secondary ventilating aids should be kept in operation at all times to guard against dangerous accumulations of gas that may develop while auxiliary ventilators are idle, and to prevent the uncontrolled movement of bodies of gas when the fans, blowers, or injectors are started.

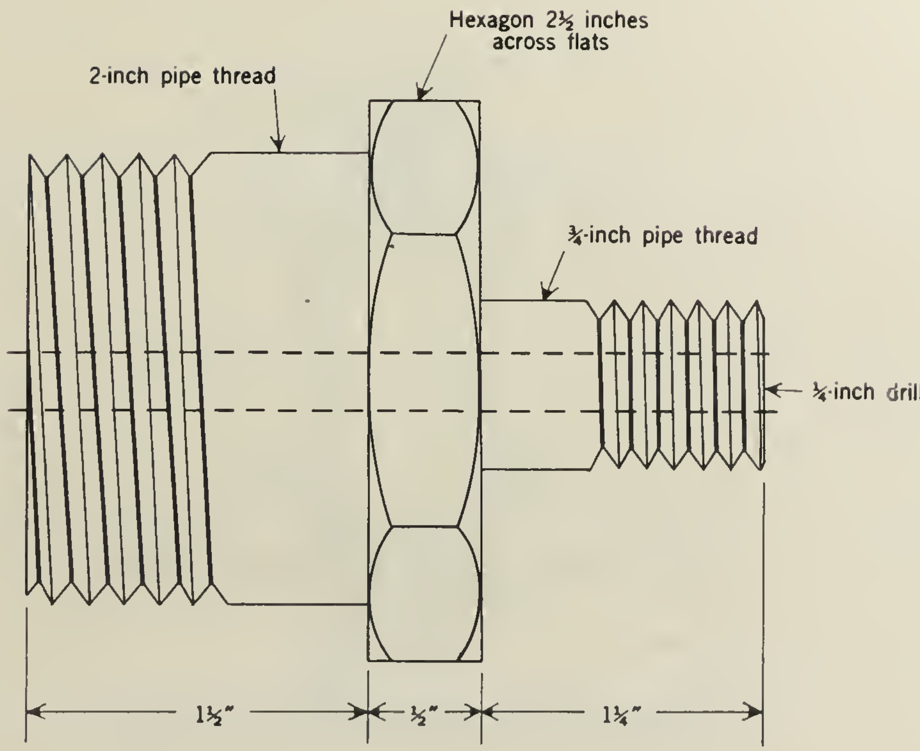

Figure 1.—Reducer for compressed-air line.

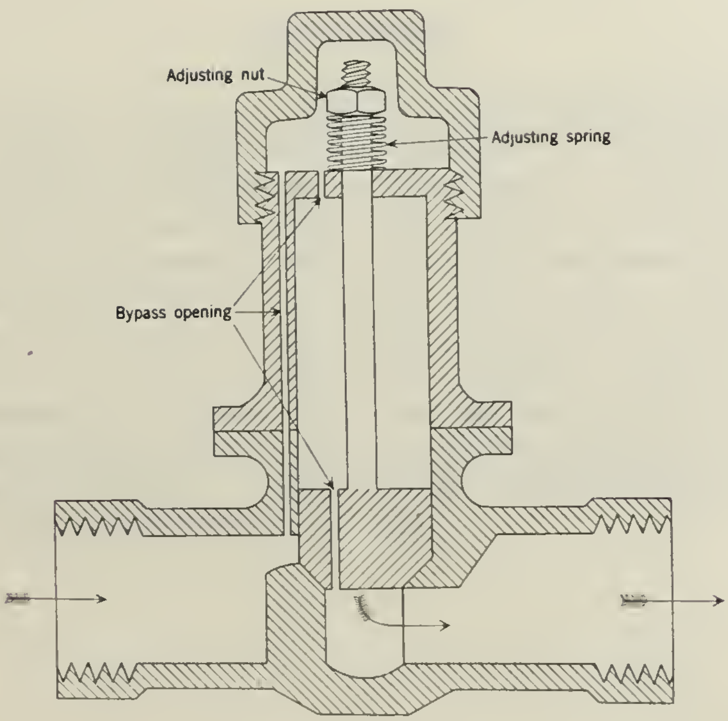

Figure 2.—Valve for compressed-air line

The use of jet installations for ventilating by compressed air affects only that portion of the working place in the immediate vicinity of the escaping air. A slight cooling effect could be felt, but rarely at a distance more than a few feet from the jet.

Compressed-air jets may be used occasionally for temporary service or in emergencies in shafts, where a number of jets fed by one air line may be placed either at the top or bottom of the shaft. Used in this manner they act as a propulsive force only. Steam jets may be used for this purpose, also, and to somewhat better advantage. Placed at the bottom of the shaft, the steam reduces the density of the upcast column of air and acts as a propulsive force. Based on fuel consumed, the efficiency of the steam jet is probably not over 15 percent, and the total efficiency of compressed-air jets used in this manner is even lower.

CONCLUSIONS

Compressed-air jets are inefficient and ineffective means of ventilating gassy mines; their use in mines of this class constitutes a confession that normal ventilation is inadequate. Conclusive evidence that compressed-air jets and blowers are hazardous is found in reports of accidents that were directly or indirectly the result of efforts to ventilate with compressed air. A few of these cases were cited in this paper, and the cost of ventilating by compressed-air jets, compared with ventilation by more effective devices or methods, proves that the management is lax at operations where their use is permitted. More progressive companies not only prohibit the use of compressed-air jets for ventilation but insert valves or controls in the lines, which prevent the miners from using them even if they desire to do so.

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DUST HAZARDS AND THEIR CONTROL IN MINING1/

By D. Harrington2/

Advocates of health and safety in mining have found themselves confronted in recent years with what might appear to be contradictory procedures with regard to dust occurrence in mines and what appear to be suitable methods or measures to give the mines and mine workers a maximum measure of protection against dust. A concerted and relatively effective drive has been made to introduce rock dust into bituminous and lignitic coal mines to prevent or limit explosion and fire disasters, while an equally determined effort has been and is now being made to curb rock dust or to eliminate it from air in metal mines to prevent or at least limit the occurrence of miners' consumption and other diseases. Both procedures are logical even though the efforts appear to be of an opposing nature.

While rock dusting has not been applied in coal mines nearly as universally or as effectively as could and should be done, nevertheless a careful analysis of the record of the past 10 years indicates that the amount of rock dusting that has been done has resulted in the saving of a minimum of 200 lives annually - certainly no small achievement. Largely but by no means wholly due to the use of rock dusting, the number of fatalities from explosions and fires in our coal mines in the last 4 years has been reduced more than 75 percent as compared with the 4-year period 1911-15, the first years of the existence of the United States Bureau of Mines, which was brought into existence in 1910 largely to try to do something about reducing the disgracefully high fatality occurrence from explosion disasters in our coal mines. Hence, rock dusting has been given at least a trial and has proved its effectiveness definitely.

The campaign to eliminate or at least to reduce the presence of rock dust in the air of metal mines has been by no means as sensational or as definitely successful as has that to introduce rock dust into bituminous and lignitic coal mines; this is due to the difference in the spectacular feature of the results of failure to apply the remedy in taking proper precautions as to air-borne dust in metal mines, as compared with failure in controlling coal-mine dust; failure in controlling dust in certain kinds of coal mines may cause scores of human beings to be hurled into eternity in a few seconds, while failure to handle metal-mine

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6954." Presented before the A.I.M.M.E., February 18, 1937.

2/ Chief, Health and Safety Branch, U. S. Bureau of Mines, Washington, D.C.

dust conditions adequately works its harm on single individuals and usually only after months or years of exposure. Notwithstanding this lack of spectacular action of air-borne rock dust, there is good reason to believe that since the beginning of the present century more underground workers in the United States have gone to their graves through the breathing of harmful dusts than have been sent there through the more sudden and spectacular instrumentality of explosions or fires. As with the coal-dust-explosion hazard, the remedies are readily available in connection with the other features of dust harmfulness (including breathing it), and organizations that really wish to avoid the various harmful effects of dust occurrence in mines can readily do it.

There are almost innumerable uncertainties connected with the harmfulness of dust, but dust is now generally believed to be detrimental to health through its effect on the throat, bronchial passages, lungs, stomach, and other internal organs as well as on the eyes, ears, nose, and other external organs and the skin covering the entire body. Many dusts (metal, mineral, and organic) are explosive, others fire spontaneously with more or less readiness under varying conditions, and practically all dusts have almost universally detrimental or so-called "nuisance" effects when they settle in the home, in the office, or on the external surfaces of buildings or on the roads, streets, or sidewalks; all of these facts (and almost innumerable others can be added) make dust one of the real scourges of present-day life, and certainly no sane person can deny that a most determined effort should be made to reduce dust or at least to confine it in such manner as to minimize its numerous harmful effects.

It now seems probable that the most harmful of the ordinary dusts to breathe is that of silica, but it must be emphasized that not even free silica (supposedly the most detrimental of the silica dusts) is harmful unless it is breathed in considerable <u>quantities or concentrations</u> and over extended periods of time; no human being ever lived any considerable span of years on this earth of ours without breathing silica dust (a large part being free silica), yet this does not mean that all of the people in the world now have or have had silicosis. In other words, the <u>quantity</u> of dust taken into the respiratory organs is a controlling factor in dust-respiratory harmfulness, and common sense indicates that the quantity of dust breathed is one of the dominant factors as to possible harmfulness to health from any kind of air-borne dust. What constitutes that quantity limit, no human being knows at this time, notwithstanding the fact that numerous so-called "threshold" limits are now being announced, some of them being embodied in State regulations or laws. Predicating dust-respiratory harmfulness on a sliding scale in proportion to the free-silica content of the dust is anything but logical; in the first place, no one knows whether a dust of 1 percent silica is or is not more harmful than a dust of 2 percent, 3 percent, or even 30 percent. Moreover, air with a certain number of dust particles per cubic foot with a silica content of 1 percent (or any other percentage) would probably be far more harmful to one person than to another or would be far more harmful to a person working at maximum capacity (as when on a contract basis) than to one on a day-pay basis, or more harmful to a person working in an atmosphere of 85° or 90° F. with

relative humidity 90 or 95 percent than to one working in an atmosphere of 60° F. with relative humidity around 60 percent. There are numerous other more or less similar contributing factors that make unworkable or futile the establishment of regulations as to the allowable number of dust particles on a sliding scale in proportion to silica content of the dust. In some industries, such as metal mining and tunneling, there may be as many different percentages of silica in the mine air as there are places working, and in many metal mines it is entirely unlikely that the percentage of silica in the air of a working place today will be at all like the percentage of silica in that same place tomorrow; to apply to most metal mines the sliding scale of allowable number of dust particles in the air in proportion to silica content would be utterly impossible of enforcement or fulfillment.

Much should be done to clarify numerous uncertainties as to air-dust harmfulness before it is attempted to establish more or less rigid air-dust standards by legal means.

At present there is definite lack of knowledge as to the size of air-borne dust likely to be harmful, and present-day so-called "standards of air dustiness" are built on numerous uncertain premises on this point alone. Different authorities find varying maximum and minimum sizes of dust in lungs of deceased silicotics ranging from as small as 0.2 micron to as large as 10 microns, and from this fact it is deduced that, as far as lung harm is concerned, the dust to be avoided is that from 2/10 to 5/10 micron, on the low side, to 6 or even as high as 10 microns, on the high side; this leaves much leeway, because an air-dust sample containing 10,000,000 particles (5/10 to 6 micron size) per cubic foot of air might easily have 30,000,000 particles 2/10 to 10 microns in size. Moreover, if the solution theory of dust harmfulness to the lungs is correct, there is the best of reasons to think that particles much smaller than 2/10 micron are very likely to be harmful. And to hold that the only harmful dust sizes are those found in the lungs is anything but good reasoning, because it is a certainty that dusts from 10 microns to 100 or more microns in size when floating in the air are drawn into the nostrils and other protective air passages, and if in large numbers soon clog these protective agencies and passages, giving practically free unobstructed entrance of the so-called "harmful" dusts (say 2/10 to 10 micron sizes) into the lungs to perform maximum injury. It does not seem to be logical to disregard these larger sizes entirely, as is now almost universally done in the so-called "dust standards" being considered and announced. Again, the present-day instruments and methods of sampling air-borne dust and making the determinations as to number of particles present are anything but accurate or definite, even when handled by technically trained experts; and methods or instruments applied to certain industrial conditions are wholly unsuited to others. A condition as to air dustiness found in one sample in a certain place is unlikely to be anything like another sample taken immediately afterwards in the same place and in the same manner; this is true where the sampling is done instantaneously by a grab method or over a period of several minutes. In other words, the technic as to quantity

particle determinations in air dustiness is by no means definite, accurate, fair, or dependable, not only as to conditions as between different plants but even as between different time periods in the same place in any one plant. With all of these as well as numerous other uncertainties as to air-borne dust, it seems to be a travesty on justice to try to embody in laws, and regulations having the force of law, rigid standards as to air dustiness; if quantity standards as to air-borne dust are made, they should certainly be labeled tentative; and the technic of taking the air samples and making the particle determinations should be very carefully outlined and enforced, otherwise much injustice can be inflicted through inexperience or the malice of the persons making the determinations.

The remedy for harm to health from dust in mines of today is chiefly twofold - use of water to "kill" the dust and use of ventilating current of fresh air to remove or dilute air dustiness where it is not feasible to "kill" the dust by water. In metal mines, no drilling of any kind should be allowed without simultaneous use of water, and the water should be under pressure and forced through the drill steel so as to "kill" the dust as it is being formed at the face of the drill hole. Many mines resist this practice because of expense of wet-drilling equipment and of placing and maintaining water lines and water at all working faces; miners resist because they hate to do anything they have not been accustomed to doing, because the use of water entails new responsibilities, and because the water tends to wet the clothing. Many other reasons are given by owners as well as by miners why wet drilling cannot be done, but in at least 9 out of 10 mines wet drilling can and should be insisted upon.

Blasting should be done after the working shift instead of during the shift, as it has been found that next to dry drilling blasting is the worst present-day producer of dust in air in sizes and concentrations likely to be harmful to those who breathe it. Moreover, there is good reason to suspect that the gases produced in blasting have a rather serious, harmful effect on those who breathe fine dusts, due to inflammation of the respiratory organs by the breathing of those gases even in relatively minute proportions. Various available adjuncts to blasting, such as water blasts or curtains, mist projectors, etc., as well as frequent wetting of the muck piles after blasting and of the face region before blasting are now required and utilized by careful mine operators, even when blasting is done at the end of the shift or on the off shift.

In coal mines, the mining machines should be equipped with water spray, and to be really effective the water should be piped to the face and be under pressure; here, again, the operator says it is impracticable, expensive, etc., but the answer is that water lines are being kept at working faces in extensive, up-to-date coal mines in several States and at a large number of faces in many metal mines, also. Water should be used on the drilling machines in metal mines and on the mining machines in coal mines, and if piped to the faces, can and should be used by the worker to wet down the broken rock or coal, together with surrounding walls, timber, floor, etc., where dust would otherwise accumulate; this is being done now in numerous mines in several States. It is due largely to the much better

ventilation of coal mines than of metal mines that coal miners are less afflicted by dust disease than are metal miners. It is also due to some extent to the better ventilation of coal mines that it is feasible to introduce rock dust into coal mines to prevent or limit explosions and fires and yet not unduly endanger the health of workers, because currents of fresh air remove floating particles of dust and few, if any, dusts that occur in coal or metal mines are harmful to health unless they are present fairly continuously in finely divided form and in large amounts in air breathed by workers through the working shift.

The general, although not universal, practice in some coal-mining regions (and a good health and safety practice) demands use of water at working faces to "kill" dust as it is being formed and rock dusting of the open parts of the mines away from the faces. Most mines rely very little on the use of water, and where rock dust is used the intent is to carry rock dusting virtually to the faces; this rock dusting to the face appears to be poor policy (unless water is unobtainable), as it introduces additional dust at the faces where about 95 percent of the dust in coal mines is formed, thus adding rock dust to the already coal-dust-impregnated air at working faces; this is likely ultimately to bring about a very serious health problem as well as to decrease visibility very greatly, as dust in the air of coal-mining places reduces visibility 50 or more percent - a decidedly dangerous factor from a safety viewpoint, even if one should disregard considerations as to health or as to dust explosibility. Moreover, cutting coal dry throws quantities of very fine coal dust into the air, which settles later on surrounding surfaces; blasting and shoveling also throw much fine dust into the air, later to settle on walls, timbers, roof, and floor; and if 1 or 2 percent of methane (explosive gas) is present, there must, in some cases, be 80 percent or more of rock dust (almost impossible to maintain) to prevent ignition of the coal dust of some of our coal mines.

CONCLUSION

This article treats only lightly of dust hazards and has been confined chiefly to dust explosibility and to respiratory disease and almost wholly to lung affections; it gives little or no consideration to the numerous ill effects on human organs other than the lungs due to breathing more or less insoluble dust, or the harm to other internal organs (stomach, liver, kidneys, etc.), or to the nose, throat, and bronchial tubes from breathing more or less soluble or so-called "poisonous" dusts; nor does it take into consideration the numerous hazards to human beings in the breathing of organic dusts, with accompanying hay fever and kindred ills. Moreover, numerous dusts, both organic and inorganic, have harmful influences of various kinds on eyes, ears, and other parts of the body, and on the skin from external contact; and some dusts cause harm to health by being absorbed through the skin. Hence, while pulmonary disease due to the breathing of air-borne dust is unquestionably of great present-day importance, its various ramifications (pneumoconiosis, silicosis, anthracosis, or these combined with tuberculosis), while vitally important, constitute by no means the only detrimental effect to human beings in coming in contact with dust, externally or internally.

A close analysis of the status of dust in industry (and in general life outside of industry also) indicates that numerous, one might almost say innumerable, uncertainties still exist as to specific features connected with dust harmfulness, and so numerous and far reaching are these uncertainties (only a few of which have been indicated in this paper) that almost the only definite fact now available is that dust is a menace and all kinds of it likely to come in contact with human beings should be reduced to a minimum or at least be held under positive control until such time as much well-planned, well-correlated research and investigative work (field and laboratory) has been done on almost every phase of the subject of dust, and much additional information has been obtained as to harmful as well as so-called "harmless" dusts.

Numerous clear-thinking persons who have made a minute, comparatively long-time study of dust harmfulness are now expressing the view that much remains unknown as to the health harm from dust, and that about the only safe procedure is to take all reasonable measures to reduce air dustiness where men must work or live. Philip Drinker, as chairman of a Committee on Preventive Engineering in connection with air dustiness, is quoted as follows in Safety Engineering for December 1936 (vol. 72, no. 6, p. 62):

> There is no satisfactory medical answer at present to this question, but the engineer is making a bad mistake if he lets men breathe heavy dust concentrations of any material. If no other reason for dust control can be found, then one should read transcripts of some of the recent suits at common law, in which fantastic damages for alleged silicosis were granted to men who breathed dust containing little or no silica. The courts and compensation boards are not impressed with subtle distinctions between dusts with 10% and 40% quartz, especially when medical experts are reluctant to make definite statements as to the comparative significance of such differences.
>
> It would be well to realize that men working in dusty trades suffer far more from respiratory troubles of all kinds than do men who work in clean air. The evidence that excessive dustiness of any kind is harmful is beyond argument.

The Colliery Guardian, probably the leading technical coal-mining magazine in Great Britain, had an editorial on "The Removal of Dust" in its issue of October 23, 1936 (vol. 153, no. 3956, p. 777), the introductory paragraphs reading:

> The British Colliery Owners' Research Association has shown much wisdom in selecting for attack that front of the dust problem which is concerned with the causation of dust in mines and its dispersal. For the time being, the duty of ascertaining the character of the dusts that may be classed as dangerous to the worker must be left to the geologist and pathologist, and to the medical men the difficult questions involved in the treatment of the diseases arising from the inhalation of dust.

From the standpoint of those diseases, it is necessary, of course, to accept certain criteria which have been laid down by the pathologists and have been tentatively adopted by the law-givers, but the dust problem extends considerably beyond the limits of those definitions, for we may account dust as a nuisance irrespective of its influence upon health and as a predisposing cause of chest ailments, and it constitutes, as we all know, a grave danger to the mine worker on account of its explosive character. So that the mine manager is disposed to regard with favour any means within reason of reducing the quantity of dust in the workings, irrespective of its character, instead of attempting meticulous eliminations of the more dangerous elements.

These are sensible statements made by thinking people, and wise mine-operating officials of the present and of the future will be far less inclined to split hairs as to what constitutes a harmful dust and concentrate their faculties and abilities on the feasible methods of handling the dust hazard so it will be reduced to a minimum in the property and as applied to the employees. At present, the best procedure is to kill the dust, and it can be done!

I. C. 6555 March, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MILLING METHODS AT THE CONCENTRATOR OF THE WALKER MINING CO., WALKERMINE, CALIFORNIA

BY

M. R. McKenzie and H. K. Lancaster

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MILLING METHODS AT THE CONCENTRATOR OF THE WALKER MINING CO., WALKERMINE, CALIF.[1]

By M. R. McKenzie[2] and H. K. Lancaster[3]

INTRODUCTION

This paper describing the milling methods at the Walkermine concentrator, Plumas County, Calif., is one of a series being prepared by the United States Bureau of Mines on milling methods and costs in the various mining districts throughout the United States.

ACKNOWLEDGMENT

The authors wish to acknowledge their indebtedness to James O. Elton, manager of the International Smelting Co., and to Henry A. Geisendorfer, manager of the Walker Mining Co., for permission to present this paper; to H. A. Geisendorfer and J. D. Roberts, shift boss, for historical facts, and to D. D. MacLellan, geologist, for geological information.

LOCATION

The Walker mine and mill are located at Walkermine, Plumas County, in the northeastern part of California and at an elevation of 6,200 feet in the Sierra Nevada. Walkermine is about 27 miles northeast of Portola, a division point on the Western Pacific Railroad which is about 60 miles west of Reno, Nev.

Weather conditions and heavy snowfall usually make the road between Portola and Walkermine impassable during the winter and early spring months. During this period, transportation of passengers and supplies is conducted over an aerial tramway 9 miles long the terminal of which terminal is located at Spring Garden, Calif. Construction of this tramway, which was built chiefly for the shipment of concentrates, was completed in October, 1920.

GEOLOGY

The geology of the Walker mine deposit is described in detail by J. S. Diller in U. S. Geological Survey Bulletin 353, 1908, Geology of the Taylorsville Region, Calif.

In brief, the orebodies occur along a shear zone which cuts through a highly garnetiferous schist, known locally as the Robinson schist. The ore shoots, which are composed of chalcopyrite in quartz and silicified schist, are not connected and may be regarded as individual deposits. They range from 300 to 2,000 feet in length and from 5 to 100 feet in width.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6555."

2 One of the consulting engineers, U. S. Bureau of Mines, and mill superintendent, Walkermine concentrator.

3 One of the consulting engineers, U. S. Bureau of Mines, and assistant mill superintendent, Walkermine concentrator.

ORE TREATED

The Walker ore has complex mineralogical associations but may be classified as a gold and silver bearing chalcopyrite-magnetite-quartz ore. Recent examination of the ore by R. E. Head, chief microscopist of the United States Bureau of Mines, showed the approximate percentages of mineral constitutents to be as follows:

	Per cent
Quartz	75.0
Garnet	5.0
Chlorite	2.5
Other nonopaque minerals	5.0
Metallic minerals	12.5
Total	100.0

The metallic minerals, which as stated amount to 12.5 per cent of the ore, are distributed as follows:

	Per cent, by weight	
Magnetite	59.76	
Pyrite	7.80	
Pyrrhotite	1.94	
Minor metallic gangue minerals	1.10	
Total metallic gangue minerals		70.60
Chalcopyrite	24.17	
Chalcocite	1.60	
Minor copper bearing minerals	2.55	
Noncopper-bearing minerals	1.08	
Total metallic ore minerals		29.40
All metallic minerals		100.00

The ore is very resistant to crushing and fine grinding. During the spring months considerable difficulty is encountered in crushing operations due to the moisture content in the ore.

The tabulation which follows presents a chemical analysis of typical mill heads averaged for a 6-month period.

Typical analysis of mill heads

Per cent					Ounces per ton	
Copper	Iron	Sulphur	Lime	Insoluble	Gold	Silver
1.687	9.0	2.1	1.1	79.2	0.05	0.833

HISTORY

Milling operations began in June, 1916, with the completion of a 75-ton capacity pilot plant, which was erected at a distance of 4,700 feet from the shaft and at a much lower elevation. Transportation of the ore from the mine to the mill was accomplished by an aerial tramway.

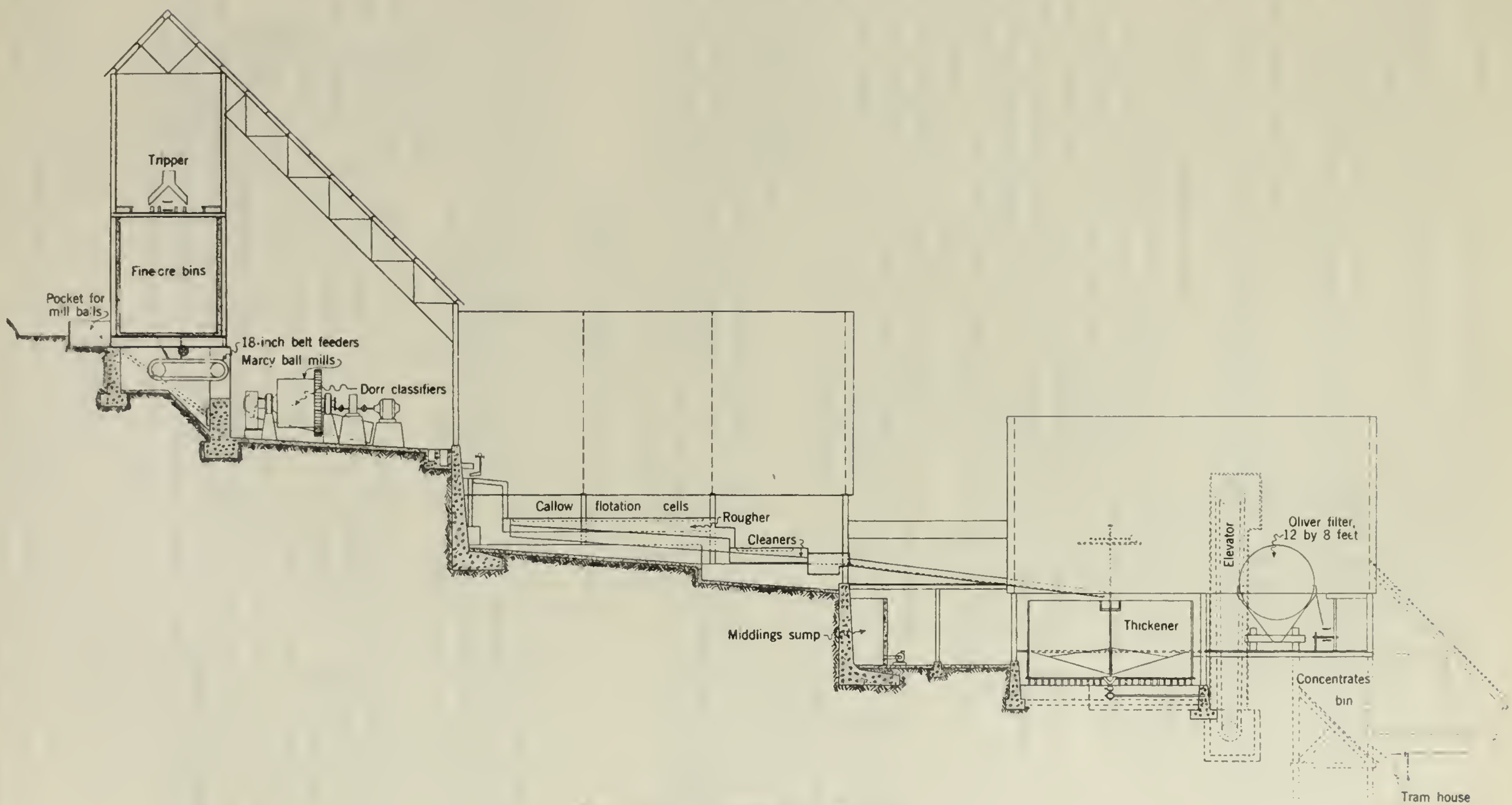

Figure 1.—Section of the Walker mine concentrator building

The treatment of ore in this pilot plant may be briefly summarized as follows:

(a) The mine ore was crushed to 2-inch size by Joshua Hendy crushers, one 12 by 24 inch and one 9 by 15 inch.

(b) The crusher product was ground to 4 per cent plus 48-mesh in two No. 64½ Marcy ball mills, each operated in closed circuit with a 4-foot 6-inch by 15-foot 9-inch Dorr classifier.

(c) The classifier overflow pulp, which contained 42 per cent of solids, was elevated a distance of about 60 feet to the flotation division, comprised entirely of sloping bottom Callow cells.

(d) The flotation concentrates were partially dewatered by two Callow cones, further thickened in a Dorr thickener and filtered by one 6 by 8 foot Oliver filter.

(e) The tailings were discharged into a canyon and no attempt was made to recover water except that contained in the concentrates thickener overflow.

Power was furnished locally by a steam plant of 350-hp. capacity until 1917 when electric power was made available by the completion of a power transmission line 14 miles long by the Great Western Power Co.

Reagents used in the pilot mill were lime, coal tar, and pine oil. Coal tar was replaced by thiocarbanilide in 1922.

During the operation of the pilot plant the International Smelting Co. acquired an interest in the property and, in addition to securing electrical power previously mentioned, drove a working tunnel 1 mile long for the purpose of developing the mine 1,000 feet vertically below the outcrop. This tunnel was also connected with the upper workings and since 1922 has served for transporting mine ore to the mill, the aerial tramway formerly used for this purpose being abandoned.

In 1922 the size of ball-mill feed was reduced from 2 inches to 1 inch by the installation of rolls. At this time the mill capacity was increased to about 300 tons of ore per day.

A larger mine production necessitated the building of a new mill of 750-ton capacity, which was completed and started to operate in December, 1923.

GENERAL DESCRIPTION OF PRESENT MILL

The present mill is built on a hillside a short distance from the portal of the main working tunnel. The location selected was well suited to standard mill construction and provided reasonable fall for gravity flow of pulp through the mill and down the valley to the tailings pond. The mill buildings are of steel and concrete with sides and roofs made of Anaconda corrugated zinc.

A typical longitudinal section of the concentrator building is presented in Figure 1.

The coarse and intermediate crushing plants are located in separate buildings at one side of the mill proper. The mill proper includes four sections. Each section is equipped with a Marcy ball mill which operates in closed circuit with a Dorr classifier; these are followed by rougher, cleaner, and scavenger Callow flotation units. A thickening and filtering unit handles the combined flotation concentrates from the four mill sections.

On account of the isolation of the mine and mill, it is necessary to maintain well-equipped machine, electrical, blacksmith, and carpenter shops. These shops can meet almost any emergency, which insures continuity of operations.

Capacity

The mill as first designed, with three No. 75 Marcy ball mills and three flotation sections, was operated for some time at the rate of 750 tons of ore per day. Grinding during

this period was maintained at 4 per cent plus 48-mesh size. The mill capacity was later increased to 1,200 tons per day by changing the degree of grinding previous to flotation treatment from 4 to 14 per cent plus 48-mesh size. It might be interesting to note that the small sacrifice in recovery entailed by coarser grinding was more than offset by the lower cost of milling which resulted from the larger tonnage treated.

In September, 1929, the capacity was further increased from 1,200 to 1,700 tons per day by the installation of a fourth mill section. This latter section is identical with the three original sections except that the grinding unit is larger and comprises a No. 77 Marcy ball mill followed by a suitable classifier; the larger grinding mill accounts for the additional 100-ton ore capacity of this section.

Water Supply

Fresh water is obtained from springs and from the mine. The spring-water supply amounts to approximately 100 gallons per minute but this water is only available for mill use after camp requirements are satisfied. Water is pumped from the mine at the rate of about 300 gallons per minute, and this source provides the chief new water supply for the mill. The only water reclaimed from milling operations is that from the concentrates dewatering division. This amounts to approximately 60 gallons per minute and is returned to the supply tank located above the mill, where it is mixed with the fresh water.

Power

The Pacific Gas and Electric Co. furnishes power to the mine and mill transformers from Caribou through its Veramont substation at 22,000 volts. Motors of 100 hp. or larger are operated at 2,200 volts and smaller motors at 440 volts. A 110-volt circuit is used for lighting.

PRESENT METHOD OF CONCENTRATING

A flow sheet of the crushing plant and concentrator with a legend which gives details of machines used is presented in Figure 2.

Coarse Crushing

Longitudinal sections of the coarse and intermediate crushing units are given in Figure 3.

Ore is hauled from the mine to the crushing plant during three shifts. Ore trains which comprise eleven 3¼-ton capacity side dumping cars are drawn by 35-hp. Baldwin-Westinghouse electric locomotives. The ore is dumped onto a sloping steel rail grizzly having 11-inch spaces and after passing through the grizzly falls into a 1,500-ton capacity, cylindrical, steel receiving bin.

Ore is drawn from this bin by a motor-driven, 42-inch, Anaconda-type pan conveyor which discharges onto an inclined grizzly having 1½-inch spaces. The oversize is fed to a Traylor crusher set at 3½ inches. The crusher is 15 by 24 inch size, is driven by belt from a 150-hp. motor and will handle 85 tons of material per hour.

The grizzly undersize drops directly onto an inclined 20-inch conveyor belt and provides a cushioning layer for the crusher product which drops onto the same belt. This belt is driven by a 15-hp. motor and delivers the ore to the intermediate crushing unit passing under an electromagnet enroute for the removal of tramp iron. A picker is stationed at this conveyor for the removal of wood.

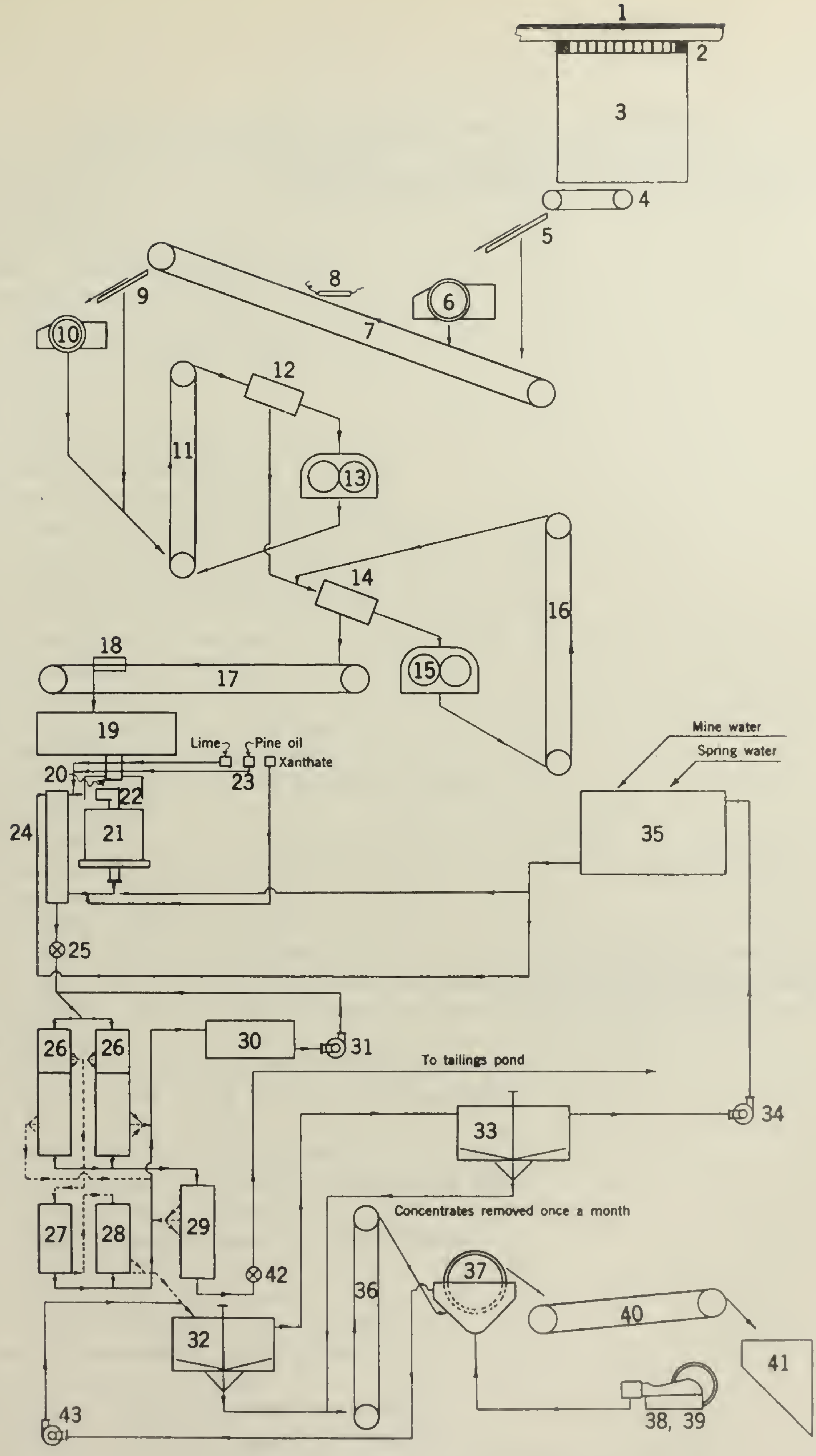

Figure 2.— Flow sheet of crushing plant and concentrator

Legend for flowsheet of crushing plant and concentrator

Number[1]	Description of machines
1	Track from mine.
2	One grizzly, 11-inch spaces.
3	Ore bin, 1,500-ton capacity.
4	One pan conveyor, 42-inch.
5	One grizzly, 1½-inch spaces.
6	One Traylor jaw crusher, 15 by 24 inch.
7	Belt conveyor, 20-inch.
8	Electromagnet.
9	One grizzly, 1¼-inch spaces.
10	Two Anaconda jaw crushers, 8 by 20 inch.
11	One bucket elevator, 16-inch.
12	One trommel screen, 1¾-inch holes.
13	One set Anaconda rolls, 24 by 55 inch, set at 1-inch.
14	Two trommel screens, 1-inch holes.
15	One set Anaconda rolls, 24 by 55 inch, set at ⅜ inch.
16	One bucket elevator, 16-inch.
17	One belt conveyor, 20-inch.
18	One tripper conveyor.
19	Fine-ore bin, 2,400-ton capacity.
20	Four feed conveyor belts, 18-inch.
21	Four Marcy ball mills, three No. 75 and one No. 77.
22	Four scoop boxes.
23	Reagent feeders.
24	Four Dorr duplex classifiers, three 6 feet by 18 feet 4 inches and one 6 feet by 23 feet 4 inches.
25	One Galigher sampler for heads.
26	Eight Callow rougher flotation machines, two 3-foot 11-pan units per mill section.
27	Four primary Callow cleaner machines, one 3-foot 4-pan unit per mill section.
28	Four secondary Callow cleaner machines, one 18-inch hoppered-type unit per mill section.
29	Two Callow scavenger machines, 3-foot 7-pan units.
30	One middlings sump, tank capacity 2,000 gallons.
31	Three Krogh pumps, 3-inch.
32	Two Dorr concentrates thickeners, 12 by 25 foot.
33	One Dorr thickener, 11 feet 4½ inches by 46 feet.
34	One centrifugal pump, 3-inch.
35	Water storage tank, capacity 100,000 gallons.
36	One bucket elevator, 16-inch.
37	One Oliver filter, 12 by 8 foot.
38	One Oliver vacuum pump, 14 by 8-inch.
39	One Oliver compressor, 9½ by 8 inch.
40	One belt conveyor, 18-inch.
41	Concentrates storage bin, 290-ton capacity.
42	One Galigher automatic sampler for tailings.
43	One centrifugal pump, 1½-inch.

1 Refer to numbers of Figure 2.

Intermediate Crushing

Ore from the coarse-crushing unit is delivered onto an inclined grizzly having 1¼-inch spaces. The grizzly oversize is fed to two 8 by 20 inch Anaconda-type jaw crushers. The grizzly undersize joins the crushed material and the combined products are fed by a bucket elevator to one 40 by 72 inch trommel screen having 1¾-inch diameter round holes. The elevator is driven by a 20-hp. motor and is equipped with buckets 16 by 8 inches in size. The trommel operates in closed circuit with a pair of 55 by 24 inch Anaconda rolls set with a 1-inch spacing; the rolls product returns to the elevator which feeds the trommel. The trommel undersize is distributed to two 40 by 72 inch trommels having 1-inch round holes. These trommels operate in closed circuit with one pair of 55 by 24 inch Anaconda rolls, set at ⅜ inch, the rolls product being returned to the trommels by a 16-inch bucket elevator.

The trommel undersize product, minus 1-inch size, comprises the feed to the grinding units and is conveyed to the 2,400-ton capacity fine-ore bin by a 20-inch belt equipped with a tripper.

The coarse rolls are driven by a 100-hp. motor, the two Anaconda crushers and the fine rolls by a 150-hp. motor, and the three trommels by a 10-hp. motor.

Grinding and Classifying

As previously indicated the concentrator is divided into four units for grinding and flotation operations. The three original grinding units are each equipped with one No. 75 Marcy ball mill which operates in closed circuit with a Dorr duplex classifier 6 feet by 18 feet 4 inches in size. The fourth unit was, as previously mentioned added to increase the capacity of the original mill and is equipped with one No. 77 Marcy ball mill which operates in closed circuit with a Dorr Duplex classifier 6 feet by 23 feet 4 inches in size.

Each No. 75 Marcy mill is driven at a speed of 24 r.p.m. by a 200-hp., 900 r.p.m., induction motor through a Falk herringbone-gear speed reducer. The No. 77 mill is also driven at a speed of 24 r.p.m. by a 200-hp., 900 r p.m., synchronous motor through a Westinghouse-Nuttal speed reducer.

Ball charges carried in the Nos. 75 and 77 mills weigh 9 and 13 tons, respectively, and are maintained by the daily addition of 4-inch forged steel balls. Ball consumption is 2.074 pounds per ton of ore ground.

Shell and feed end liners are of manganese steel, the shell liners being of the ship-lap type. Grate sections are of rolled chrome steel and have ¼-inch openings. The tabulation which follows gives the life of liner parts and the consumption of liners per ton of ore ground.

Life of liner parts and consumption of liners per ton of ore ground

	Life of liner, hours	Liner consumption, pounds per ton of ore
Shell liners	3,200	0.273
Feed end liners	4,300	0.176
Grate bars	4,200	0.176
Total		0.563

The classifiers used with the No. 75 grinding mills are set at a slope of 3½ inches to the foot and are operated at a speed of 27 strokes per minute by 5-hp. motors. The classifier which serves the No. 77 mill is set at a slope of 3 inches per foot and is operated at a speed of 25 strokes per minute by a 5-hp. motor.

Ore from the fine-ore bins is delivered to the center of each drum and scoop type ball-mill feeder by an 18-inch conveyor belt. Each No. 75 mill receives feed at the rate of $16\frac{2}{3}$ tons per hour and the No. 77 mill at the rate of $23\frac{1}{3}$ tons per hour. The grinding mills operate with an average circulating load of 145 per cent and with pulps containing from 77 to 78 per cent of solids.

The degree of grinding is maintained at 12 per cent plus 48-mesh; classifier overflow pulps contain from 46 to 50 per cent of solids.

Table 1, page 9 presents screen analyses of ball-mill feed and of intermediate and final grinding-circuit products.

Flotation

For simplicity of control and metallurgical accounting the classifier overflow pulps of the four grinding sections are combined, sampled and then distributed equally to four flotation units. The equipment of each flotation section comprises two 3-foot, 11-pan, standard Callow rougher machines, one 3-foot, 4-pan, standard primary Callow cleaner, and one 18-inch, hoppered-type, secondary Callow cleaner.

The two rougher machines operate in parallel; rougher concentrates are removed from the first three pans and middlings froths from the remaining eight pans. The rougher concentrates, which have an average content of 15 per cent of copper, are cleaned in two stages, the final stage producing finished concentrates. The middlings froths of the rougher machines join the tailings of the two cleaner cells and flow by gravity to a common pump sump, and from there are returned to the head of the rougher cells by three 3-inch Krogh sand pumps, each of which is driven by a 15-hp. motor.

The tailings from the rougher cells of the four flotation sections are combined and distributed to two 3-foot, 7-pan, Callow scavenger units which produce middlings froths and final waste tailings. The middlings froths join the middlings of the rougher and cleaner units in the common sump and are returned to the heads of the roughers.

Cell blankets of 4-ply, quilted, 18-ounce canvas are used and have a life of from 60 to 90 days.

Air for flotation operations is furnished at 4.15 pounds per square inch pressure by two No. $6\frac{1}{2}$ Roots bloweres and one Connersville blower. The Roots blowers are each link-belt driven at a speed of 172 r.p.m. by a 150-hp. motor. The Connersville blower is also link-belt driven at a speed of 240 r.p.m. by a 75-hp. motor.

The reagents used in flotation comprise lime, potassium ethyl xanthate (Z-3), and steam-distilled pine oil. Sodium aerofloat was used for a short period; but recently its use has been discontinued.

Dry hydrated lime is fed to the ball mills from small hoppered feeders at the rate of 1.2 to 1.6 pounds per ton of ore milled. These amounts produce a protective alkalinity of about 1.09 pounds of CaO per ton of mill water which is sufficient to insure the desired metallurgical results. Operators are required to make hourly titrations for protective alkalinity and to make immediately any changes in the rate of adding lime as indicated by these titrations.

Xanthate is added as a 25 per cent solution to the classifier overflow pulps by a scraper feeder at the rate of 0.18 pound per ton of ore treated.

Steam-distilled pine oil is added to the ball mills by a scraper feeder at the rate of 0.16 to 0.22 pound per ton of ore. A rather large amount of this reagent is necessary on account of the coarseness of the flotation feed and also due to the fact that flotation operations are conducted in a circuit which is essentially a fresh-water circuit.

DEWATERING AND HANDLING OF CONCENTRATES

The concentrates pulps which contain from 20 to 25 per cent of solids, flow by gravity to two 25 by 12 foot Dorr thickeners. The thickened pulps containing 75 per cent of solids are delivered to one 8 by 12 foot Oliver drum-type filter by a 16-inch bucket elevator. The filter is chain driven by a 3-hp. motor. The concentrates produced are handled by operating the filter on two of the three daily shifts, one operator being required on each shift.

A vacuum amounting to 22 inches of mercury is maintained at the filter by one 14 by 8 inch Oliver vacuum pump which is driven by belt from a 15-hp. motor. Blowing air is furnished at 5 pounds pressure by a 9½ by 8 inch Oliver compressor, driven by belt from a 15-hp. motor. The filtrate is handled by a 1½-inch centrifugal pump in place of the usual barometric leg. A filter cover gives approximately six months of service before replacement is necessary.

The overflows from the two Dorr thickeners are conveyed to a spare 45 by 12 foot thickener, where a small additional recovery of fine concentrates is made. These concentrates are allowed to accumulate and are dewatered in the filter about once each month.

The filter cake, which averages 1 to ½ inch in thickness and which contains from 8 to 10 per cent of moisture, is discharged onto a 14-inch conveyor belt and delivered by this belt to a 290-ton capacity storage bin. The concentrates are loaded from this storage bin into 800-pound capacity tramway buckets; the latter are trammed to Spring Garden where the concentrates are dumped into railroad cars for shipment to the Tooele plant of the International Smelting Co.

The tramway is of the double rope type and is 9 miles long. The loaded side is equipped with 1½-inch locked-coil track cable and the light side with 1-inch cable of the same construction. The buckets are equipped with automatic grips which engage a ¾-inch Lang Lay traction rope which is driven by a 50-hp. motor.

Labor employed on the tramway includes 1 foreman, 3 loaders, 3 unloaders, 2 line riders, and 1 agent, who is located at Spring Garden.

Since the completion of the aerial tramway in 1922, it has been an important factor in plant operation as it is the only means of transporting passengers, mine and mill supplies and camp provisions during the months of heavy snowfall.

DISPOSAL OF TAILINGS

The tailings of the scavenger flotation cells contain from 33 to 36 per cent of solids and are conveyed by a wooden launder for a distance of about ¾ mile to a large impounding pond. Proper precautions are taken to prevent tailings from entering near-by streams.

MILL SAMPLING

Samples of the heads as represented by the combined classifier overflow pulps, the concentrates, and the final tailings are taken during each shift by Galigher automatic samplers. The samples are dewatered in a small pressure filter and after being dried are split to convenient-size assay pulps.

An analysis for copper content is made on each shift sample for mill guidance and control. A composite sample is made from shift samples for metallurgical accounting. These composite samples are assayed for copper, gold, silver, and insoluble contents.

Cars of concentrates are sampled by pipe samplers at Spring Garden before being shipped to the smelter.

METALLURGICAL AND OPERATING DATA

Screen analyses of concentrator intermediate and final products are presented in Table 1. Table 2 gives chemical analyses of mill heads, final concentrates, and tailings; and Table 3 shows the percentage distributions of copper, silver, gold, iron, and insoluble in final concentrates and tailings. Metallurgical data for the period June to November, 1930, are presented in Table 4, and the distribution of labor is shown in Table 5.

Table 1.- Screen analyses of concentrates intermediate and final products

Screen size	Weight, per cent									
	No. 75 Marcy mill units				No.77 Marcy mill unit			Flotation products		
	Ball-mill feed	Ball-mill dis-charge	Clas-sifier sands	Clas-sifier overflow	Ball-mill dis-charge	Clas-sifier sands	Clas-sifier over-flow	Concen-trates	Middlings	Tail-ings
Plus 1-inch	5.4	-	-	-	-	-	-	-	-	-
Plus ½-inch	41.3	-	1.4	-	-	0.9	-	-	-	-
Plus 4-mesh	23.0	2.6	8.1	-	1.2	4.7	-	-	-	-
Plus 8-mesh	9.5	4.7	8.5	-	2.5	6.0	-	-	-	-
Plus 30-mesh	9.0	30.0	43.5	2.5	23.8	46.5	2.2	-	-	3.2
Plus 48-mesh	1.8	14.0	14.8	11.2	14.3	15.4	10.5	4.8	3.0	15.7
Plus 65-mesh	2.9	8.1	6.4	22.4	11.4	8.4	14.6	6.8	1.5	5.5
Plus 100-mesh	0.5	6.8	3.3	2.5	7.0	3.3	6.2	18.1	4.5	13.8
Plus 150-mesh	2.4	6.0	3.8	15.5	10.5	4.2	9.0	14.5	4.4	10.0
Plus 200-mesh	2.9	6.1	3.7	2.3	6.5	3.3	12.2	6.8	6.0	10.7
Minus 200-mesh	1.3	21.7	6.5	43.6	22.8	7.3	45.3	49.0	80.6	41.1
Totals	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Table 2.- Chemical analyses of mill products, June to November, 1930

	Weight, per cent	Analyses				
		Copper, per cent	Silver, ounces per ton	Gold, ounces per ton	Insoluble, per cent	Iron, per cent
Heads	100.0	1.687	0.833	0.05	79.0	9.0
Concentrates	6.41	24.00	9.920	0.49	14.5	22.1
Tailings	93.59	0.163	0.200	0.02	83.5	8.1

Table 3.- Mill recoveries and losses, June to November, 1930

	Weight, per cent	Distributions, per cent				
		Copper	Silver	Gold	Insoluble	Iron
Concentrates	6.41	91.2	73.3	62.7	1.2	15.8
Tailings	93.59	8.8	23.7	37.3	98.8	84.2

Table 4.- Metallurgical data, June to November, 1930

Ore treated, total	tons	268,255
Days operated	number	169.67
Operating time per day	hours	24
Amount of ore treated per 24 hours, average	tons	1,580.62
Ore treated per man-shift per 24 hours, average	do	38.51
Sections operated, average	number	3.84
Ore treated per section per 24 hours, average:		
No. 75 Marcy mill units	tons	400
No. 77 Marcy mill unit	do	500
Concentrates produced, total	do	17,290.54
Copper produced, total	pounds	84,027,948
Concentrates produced per 24 hours, average	tons	101.90
Recoveries:		
Copper	per cent	91.21
Silver	do	76.32
Gold	do	62.73
Ratio of concentration	tons into 1	15.57
Pressure of flotation air, per square inch	pounds	4.15
Alkalinity of mill water, CaO per ton of water	do	1.09
Plus 48-mesh material in flotation tailings	per cent	11.10
Consumptions of water, reagents, and supplies per ton of ore milled:		
Net water used	gallons	325 to 350
Lime	pounds	1.40
Pine oil	do	0.321
Potassium ethyl xanthate (Z-3)	do	0.084
Sodium aerofloat (use discontinued)	do	0.075
Balls	do	2.074
Liners	do	0.563

Table 5.- Distribution of labor

	Number per 24 hours
Mill superintendence:	
Mill superintendent	1
Shift foremen	3
Coarse and intermediate crushing departments:	
Crusher operators	2
Crusher helpers	2
Rolls	2
Screens	2
Electromagnet	2
Grinding and flotation departments:	
Ball mills	3
Flotation operators	3
Flotation helpers	3
Filter department:	
Operators	2
Repair and shop crews:	
One repair foreman and 20 men	21

Distribution of electric power,
May, 1931

Department	Kilowatt-hours	Per cent
Primary crushing	42,804	7.22
Secondary crushing	42,804	7.23
Grinding	348,631	58.83
Flotation	149,765	25.27
Filtration	8,561	1.45
Totals	592,565	100.00

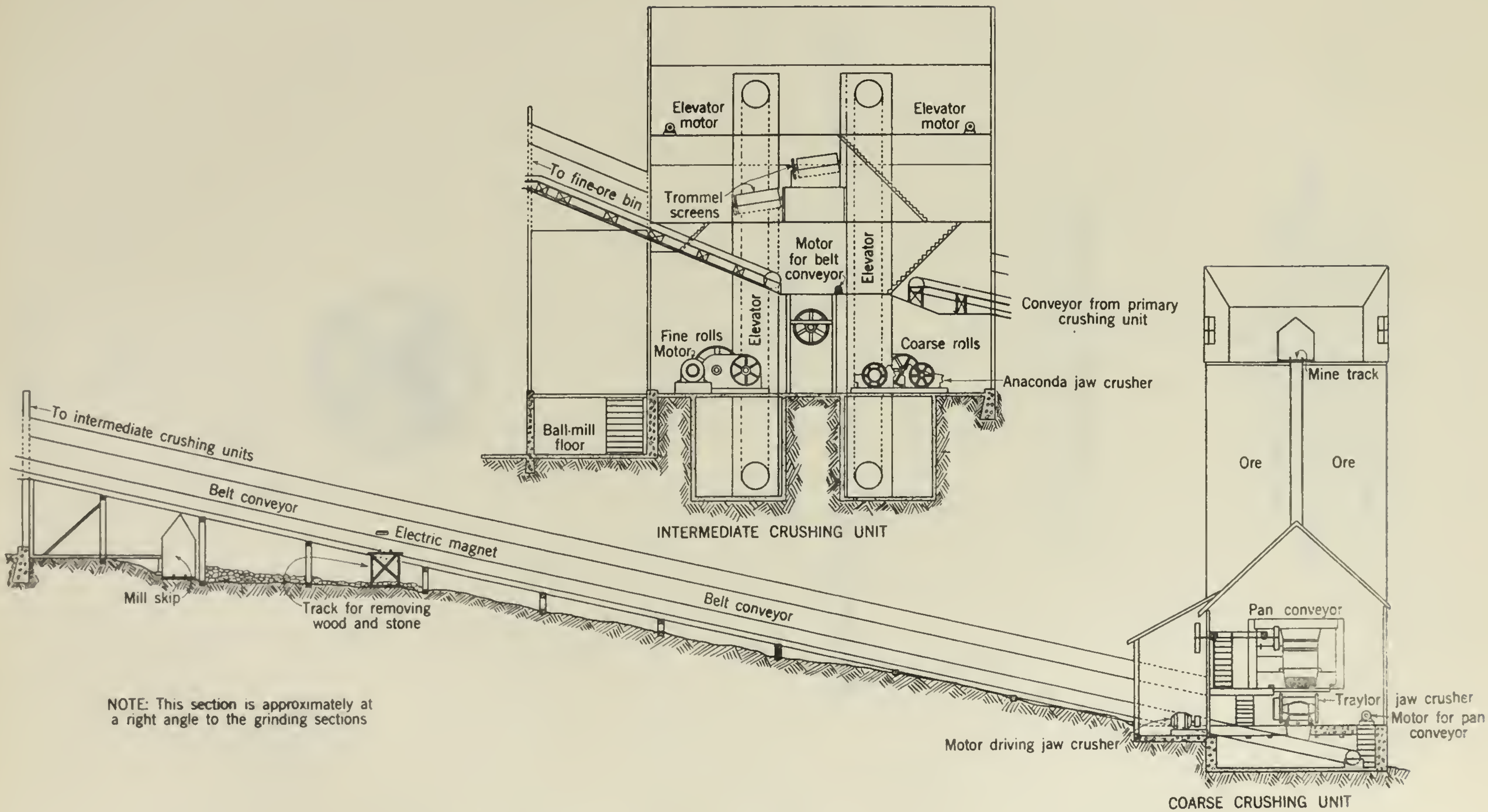

Figure 3.—Longitudinal sections of the coarse and intermediate crushing units

I. C. 6556 April, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

250 VERSUS 500 VOLTS OR MORE FOR CIRCUITS IN GASSY COAL MINES

BY

L. C. Ilsley

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I.C.6556
April, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

250 VERSUS 500 VOLTS OR MORE FOR CIRCUITS IN GASSY COAL MINES[1]

By L. C. Ilsley[2]

The maintenance of permissible equipment which is tested and inspected at the Pittsburgh Experiment Station of the United States Bureau of Mines is dependent to a surprisng degree upon the voltage at which the equipment is operated. The difficulty of maintaining 500-volt equipment is frequently brought to the bureau's attention, and it is quite possible that if the bureau had appreciated at the beginning the full hazard involved, it would never have undertaken the approval of equipment operating at voltages greater than 250 volts.

In this connection, the following statement from State mine inspection departments and other sources are of interest as showing the general feeling in safety organizations with respect to voltages higher than 250. These statements, although obtained as a result of a questionnaire sent out several years ago, still apply very well to present conditions.

Comments from State Inspection Group

A. I have not had time to consider seriously the proposed change from every angle, but looking at it from a standpoint of hazard, feel that the danger would be greatly increased. I have always occupied the position that underground voltage on portable equipment should never exceed 250 volts.

High voltage of course would mean smaller transmission lines, which would be more economical and no doubt would require in several instances smaller equipment; notwithstanding, if this statement is true, the writer is still favorable to low voltage on underground portable equipment.

B. I believe 500 volts should be the limit on the best equipment under the best conditions and that the U. S. Bureau of Mines should do all in its power to limit the voltage to 500 volts on permissible equipment, as any increase in voltage means increased hazard.

C. I will refer you to a section of our mining law, entitled "An Act to Regulate the Use of Electricity in the Mines of the State of -----" which reads in part, as follows: Trolley wires, or other exposed electrical wires, shall not carry a voltage above 275 volts.

You may readily see by this that we are not permitted to operate mining machines under ground in -----, with more than 275 volts at the terminals, except that we are permitted to install transforming stations throughout the mines, after complying with certain regulations at the mine.

However, as we have had to date, within the calendar year of 1925, six men electrocuted in the mines of the State, using a voltage of 275 volts, I am of the opinion that if this voltage was increased, the hazard would also be increased in proportion.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6556."

2 Electrical engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

D. I do not care to make a statement for publication: however, I am referring you to our mining laws. On page 27 and 28 you will find what the laws of ----- require relative to the installation of electricity in the mines.

"Hereafter where electricity is installed no higher than two hundred seventy-five (275) volts shall be used underground, except for transmission, or for application to transformers or other apparata, where the high-voltage circuit is stationary. High-voltage motors, and transformers, shall be installed in suitable chambers, built of fire-proof construction, and well ventilated, and shall not be placed on intake, or between intake and return air courses. All high-voltage lines lines and apparata must be clearly marked, to indicate their danger. Main and distribution switchboards for high voltage shall be made of insulating and incombustible material, such as marble or slate, and to be free of all metallic veins, or of material equivalent thereto. Same shall be installed in a dry place. All switches or other instruments used in connection with high voltage shall be installed on suitable switchboards, with at least a three-foot passageway in front and rear properly floored, and kept locked by a door at each end that can be opened from the inside without a key. With higher than medium voltage, there shall not be any live metal work on the front of switchboard, and the entry circuit must be protected by an oil breaker switch on each pole equipped with automatic overload trip.

"To insure safety, all conductors with higher than medium voltage shall be provided with sufficient insulation of standard manufacture and steel armored covering, or its equivalent, covering to be grounded at least every four hundred (400) feet. No grounding of phase of alternating current shall be permitted. All branch trolley lines shall be provided with automatic trolley devices to cut off the current. All joints and conductors shall be mechanically and electrically efficient. When joints are complete, they shall be insulated to the same extent as the rest of the wire. Both rails on main haulage roads shall be bonded and cross-bonded at points not to exceed three hundred (300) feet apart. All appara, such as transformer cases or switch cases, generators or motor frames and pipe work on rear of switchboards, shall be properly grounded.

"By low voltage herein is meant a voltage of less than one hundred ten (110) volts. By medium voltage herein is meant a voltage ranging from one hundred ten (110) volts to two hundred seventy-five (275) volts. By high voltage herein is meant a voltage abcut two hundred seventy-five (275) volts."

E. The highest voltage allowed in ----- mines is 275 volts, so this is considerably below the amount you mention in your letter.

F. From my experience in the last 25 years in coal mines in the United States., I am of the opinion that in no case should the voltage enter a machine terminal in the excess of 220 volts A.C., and 250 volts D.C.

A contact with 500 volts in a coal mine where it is damp, ninety-nine times out of a hundred proves fatal, due to the grease and dust. It would be impractical to have higher voltage in a mine due to the chances of having a short from the dust and grease around underground machinery.

On the surface where there is more room around the machines and a dry insulated platform, I would have no objections to 500 volts, but I do think it a menace to employees to have high voltage enter the terminal in underground machinery.

G. As a result of some years in the Southwest, where the first high-voltage equipment for operating mining machines was installed in 1912, I have had some experiences which have given me decided views on this subject. So far as I know we have not in this State any A.C. installations for operating mining machines and

I trust that we will not have any in the future. In the McAlester-Wilburton field and the Henryetta field in Oklahoma many of the mines were equipped with 440 A.C. installations, and in the Henryetta field in particular there have been quite a number of fatalities due to underground employees coming in contact with the wires or with charged mining machines. If A.C. is underground for operating mining machines, it should not be used in excess of 275 volts. This is my opinion.

With reference to the particular case cited in the letter accompanying your letter of December 19, presumably the equipment of approved mining machines has passed the test at 500 volts. If this is true, if current is furnished to this machine in excess of 500 volts then the machine is not an approved machine: that is, it is not permissible. Manufacturers of permissible explosives furnish samples to the bureau, which samples are tested by the explosives section and the sticks of permissible explosive are of certain length and certain diameter, and the manufacturers are not permitted to deviate from this size, since it is the size established by the tests as being safe. For like reason the equipment of mining machines should be held to a certain standard voltage, and no deviation should be permitted from this voltage. This is my opinion in the matter.

I think it is extremely dangerous to boost the voltage as high as 600 or 700 volts on a 500-volt machine. I am not well enough informed in the matter to be able to speak with authority, but it seems to me that this would result in injury to the machines just as much as feeding too low a voltage to a D.C. mining machine results in injury to the armature.

In brief, so far as this bureau is concerned we are opposed to such practices, and we want to go on record as opposed to the installation of A.C. current for operation of mining machines underground.

H. As the voltage increases, the danger increases also. Think maximum voltage to equipment should not exceed 500.

I. If a machine is equipped for 500 volts, any higher voltage is an overload, and will either burn or blow the fuse, thus causing flames, which as we all know is a great hazard in coal mines.

J. I would in discussing this matter say that a piece of permissible equipment designed to carry a pressure or potential of 500 volts, be rendered unsafe and that the hazard would naturally be increased, if a higher voltage is used than the equipment is designed for.

It has been the coal industry's experience in ----- that high voltage underground increases the hazards.

This fact has been recognized by the legislature in -----, by the enactment of a statutory provision of law that limits the voltage used underground to 325. The mines that carry a higher voltage are those that were in operation prior to the passage and approval of the act above mentioned.

It is, therefore, the opinion of the writer that higher voltage used underground, whether used in connection with permissible equipment or otherwise, can only lead to greater hazard.

K. The question of increasing to 600 or 700 volts the voltage of circuits furnishing power to 500-volt equipment has been taken up with several electricians connected with the mining industry, and it seems to be the general opinion that such practice should not be approved.

Any increase in voltage over 500 tends to burden the devices provided for safety.

L. I am very much of the opinion that it would be the unsafe thing to do in the coal mines of this State to increase the voltage to more than 500 volts. In fact, most of our mines that are equipped with electricity only 250 volts, which is very satisfactory to those at interest. Very few, however, use as many as 500 volts and I do not believe that they could be encouraged to use a higher voltage than this.

The seams of coal worked in this State, where mining machines are used, generally speaking, are thin. Therefore, in my opinion, it would be unsafe to increase the voltage to more than 500 volts, as the top is low, and besides, there is considerable water in these mines.

I am not so sure at this writing whether the Division of Mines would have authority, under our mining laws, to issue an order prohibiting the use of more than 500 volts in a coal mine, but since we have most all other authorities, I can see no reason why we should not have this authority, and in the event that some one should suggest increasing the voltage at their mine to more than 500 volts, I believe that it would be entirely proper for the Division of Mines to prohibit it by way of an order from this department, which I am sure would be obeyed, in case we should do so, as the mine operators throughout this state are exceedingly courteous and anxious to carry into effect any and all orders issued by this department.

M. In general, I do not believe that the voltage on such machinery as haulage locomotives and mining machines should be raised above 500 volts. It is my opinion that such machines should not use over 250 volts. If voltage is increased from 500 to 700 it will only be a matter of time until there will be a tendency to raise it to 800 or 1000 volts.
A 500-volt machine will give better service than machines of higher voltage, and 500 volts will cause much less trouble with short-circuiting, less sparking at the commutators, etc.

In case a machine is not properly grounded, as a haulage locomotive having a large amount of sand or dirt between the rails and the wheels, there is a much greater chance to electrocute a workman on a 700-volt than on a 500-volt machine. It is commonly accented that 0.25 amperes is sufficient current to cause electrocution. Raising the voltage from 500 to 700 volts with the resistance which would be practically equal with both voltages, the hazard is greatly increased. It will probably be stated that 500 volts are generally sufficient to electrocute. In general, this is the case but at times it is not. In metal mines, which in most instances are hot and wet, 250 volts are sufficient to cause death due to the low resistance.

Underground electrical machinery does not get the same care as that on the surface; trolley lines in general are not out of reach of the workers, machine cables become more or less worn, all of which should point to reducing rather than increasing the underground voltage.

N. The position of the Department of Mines through the Bureau of Labor and Industry is best evidenced by the following amendment which it had added to Section 1879 of the Mine Laws of -----, in cooperation with the coal operators of this State at the regular session of the General Assembly in 1924 "Electric power used on trolley wires in collieries established after July first, nineteen hundred and twenty-four, shall not exceed three hundred volts.

It is our opinion, and apparently that of the coal operators of -----, that a voltage of over 300 is not necessary for the successful operation of coal mines, and that above the 300 voltage, unnecessary hazards are created. We hope that

your bureau will not approve any underground electrical equipment for over 500 volts, and that when possible will make it even less than that.

O. Protection of wires; voltage on trolley wires in new collieries: In all electrically equipped mines at partings or other places where men are required to pass under bare power wires hanging less than 6 feet 6 inches above the place where he is required to walk, the same shall be protected, either by having the wire in a trench cut in the top, or by means of a board along the wire or by any other method that shall be approved by the State Mine Inspector or his assistant. Electric power used on trolley wires in collieries established after July first, nineteen hundred and twenty-four, shall not exceed three hundred volts. Any person violating the provisions of this section shall, upon conviction, be fined not less than ten or more than fifty dollars.

P. It is our opinion that a higher voltage than 500 is not desirable in underground workings. The darkness of the interior of mines is a distinct disadvantage from that of surface conductors of electricity where all can be plainly visible. Another disadvantage is the lack of space to put such conductors out of reach of persons traveling, particularly in carrying and using tools along the ways the conductors are strung. The liability of falls of coal and roof is also a distinct disadvantage in applying higher voltage to underground machines except as applying to stationary machine where conductors are in the form of armored cables or of metallic coverings.

The laws of this State, Sec. 146, Article XIV, reads: "No higher voltage than medium voltage shall be used underground, except for transmission or for application to transformers, or other apparatus, in which the whole of the high-voltage circuit is stationary.

In gaseous mines, high-voltage transmission cables shall be installed in the intake airway only, and high-voltage transformers shall be installed only in suitable chambers ventilated by the intake air which has not passed through or by a gaseous district.

All high-voltage machines, apparatus, and lines shall be so marked as to clearly indicate that they are dangerous by the work "DANGER" placed at frequent intervals.

Q. I am opposed to electric current in the mines in excess of 500 volts.

Comments From Manufacturers Group

A. In our opinion, there is an extra hazard created by boosting the voltage of 500-volt circuits above this figure. This may not be true to any great extent in the motor itself, but does become serious in the starting apparatus and in the cables used in wiring the various machines.

In the starting apparatus and fuse boxes, used on permissible portable machines, the amount of space available is usually very limited and arcing distances are kept at an absolute minimum. Dampness is always present in mine air; also, these machines are required to work in wet places frequently, which lowers the apparent arcing distance by lowering the dialectic strength of the air or other insulating material used in this apparatus.

The duplex cables usually used with mining machines, of different kinds is frequently the commercial standard 600-volt insulation and while it is sufficient to withstand 600 or 700 volts under normal conditions we feel that when the 500-volt circuit is raised to 600 or 700 volts the extra hazard from this cause should be prohibited.

In addition to the above, we have always felt very strongly that the advantage gained through raising this voltage above normal was more than offset by the additional danger to human life.

B. With reference to the advisability of increasing the present 500-volt circuit to 600 or 700 volts for permissible equipment we wish to state that we do not feel as though we are in a position to furnish information of any great value on this subject, as all of our permissible equipment is built for operating on not over 250 volts.

We do, however, believe that voltages above 500 would likely cause considerable trouble in mining operations, principally in permissible equipment.

All manufacturers of mining equipment are constantly facing the problem of designing their equipment with the largest maximum output into a minimum space.

This of course is not true in electrical work above ground where plenty of space is usually available and equipment can be designed to operate successfully on higher voltages.

Another element which we believe enters into this question to a great extent, is that there is more or less dampness in all mining operation which requires additional care and caution in the designing of electrical equipment.

C. As manufacturers of portable electrical tools which are limited as to space for windings and insulation, we naturally desire that the voltage be kept as low as possible within working limits. We have gradually been compelled to increase the voltage of our tools for street railway use from 500 to 700 volts. This has made necessary the use of interpoles, more commutator bars and more finely divided windings. We are now building practically all of our street railway tools for 700 volts.

We feel, however, that the hazard underground is much greater and believe that a maximum voltage of 550 is all that should be allowed. Quite a number of our coal drills of the open type are operating successfully at this voltage.

D. We have had no experience of permissible electric equipment. But on our semienclosed electric equipment, 500 volt, we are experiencing considerable trouble where the mines are using 650 and 700 volts instead of keeping within the reasonable limit the equipment has been designed for.

I sincerely hope that the Bureau of Mines will be successful in taking some steps toward making the 500 volts mines pay more attention to the voltage variation

E. It is our belief that in most mines, where 500 or 550 volt circuits are used, power is generated at 575 or 600 volts. This is to counteract the voltage drop between the generator equipment and the face. Further, even with this system the potential near the face is generally 500 volts or less. So that equipment which was intended for operation at the face and was designed for 500 volts was in general satisfactory. However, there is a certain class of electrical equipment that because of its function might be located or operated at any place in the mine and if it were located near the generating equipment, it would of course have to be suitable for operation on 600 volts.

In the last few years most of the coal companies, because of uneconomical operating conditions, at reduced voltage, have attempted by means of feeders and underground substations, which are becoming more popular, to keep a more constant potential throughout trolley and feeder circuits. This, of course, means that a greater amount of electrical equipment will be operated on a potential of over 500 volts in mining service and we feel, now more than ever, that to limit the approval of permissible equipment to 500 volts would limit the application and usefulness of the apparatus in the mine.

The standard practice of this company has been to design generating equipment for 600 volts service, rate the stationary motor and control equipment of 550 volts suitable for operation at a 10 per cent plus or minus voltage variation and rate the locomotive at 500 volts suitable for operation on 600 volts. This, in our opinion, is necessary for as long as the power is generated at 600 volts, it will be necessary to provide equipment suitable for operation on 600 volts and we would follow this standard practice in any permissible design which we would attempt.

We do not anticipate any troubles from the electrical point of view in designing permissible equipment suitable for 600-volt operation, as our electrical problems such as space and creepage distance would be the same in this class of equipment as it is in our totally enclosed equipment and this we design for mining work suitable for 600-volt operation.

We could not recommend the use of 700-volt circuits in the mines and further we believe that this is entirely unnecessary and would bring added expense to both electrical manufacturers and the operators because of our standard existing conditions.

F. Any one advocating more than 500 volts is doing it for a selfish purpose and not an economical recommendation.

The A.C. people only have one argument and that is, they can transmit under high voltage with less expense or investment in copper. On the other hand, there is no use in cutting coal and loading it without economical haulage. There is no successful A.C. haulage locomotives, and while the man who puts in A.C. high tension for cutting has to follow up with D.C. 500 volts for haulage. The difference in cost for maintaining the A.C. mining machine in copper more than offsets the additional cost of copper for 500 volts D.C., considering that it is necessary to use D.C. for haulage. In other words, making one system of D.C. for cutting and hauling is cheaper and more satisfactory than maintaining the two systems. I have watched a great many operations who have both systems and in each case where they are using A.C. it is necessary to put down a number of holes and protect the cables. Such labor cost has run into more expense than the additional cost of copper to carry through sufficient voltage for the 500 D.C.

In my experience, I will frankly say that I would not recommend anything over 500 volts D.C. and if any one can show me where higher voltage is any advantage as to safety or economy, I am open to conviction.

G. In reading over the enclosed copy of part of a letter the bureau has received from a manufacturer it would seem to the writer that the manufacturer mentioned declines to undertake the expense necessary to change his present design to overcome his difficulties and is trying to induce the bureau to insist on a thing that might help him as a less expensive way out.

The writer's experience has been similar to the manufacturer's mentioned, but considerable (personal) research work and constant improvement have overcome these troubles. The trouble mentioned in the manufacturer's letter should not occur in our outfits up to 750 volts.

From a practical standpoint it is the writer's opinion that such a ruling is very unwise. In fact, we have less trouble and upkeep expense on higher voltage, and the hazard from shock is practically the same on 500 to 700 volts.

The hazard from shock could be reduced, if not eliminated, by grounding the negative wire from the reel to the machine on the machine frame and grounding the same negative wire on the reel shaft, this same wire to run out and connect to the rail in entry as at present.

The negative wire, as suggested above, is now also connected to the rail for return to generator. This all-metal negative return circuit would connect both machine and truck to the rail even if the machine was on the bottom under the coal.

Now if the machine should become charged, the current would flow out the return cable and not through the man who might come in contact with outside of machine or truck. Also, if the breakdown should be sufficient to cause a large current to flow, the fuse in the trolley clip would protect the apparatus.

Necessarily the operator must keep his cables connected correctly. Failure to do this would only blow the fuse in the trolley clip if the operator persisted in trying to connect with the trolley wire.

More attention would necessarily be paid to the correct connection of cables than is shown in the field to-day, which would promote greater safety..

H. We do not make mining machines, as the name usually implies, for use in undercutting coal. Our product is for boring the shot-firing holes and other tools for use around the mines. Therefore, our experience would not help you.

So far as we have been able to observe, we do not believe that high voltages above 500 volts are safe in the hands of the kind of men usually working in mines, especially in damp mines.

I. Beg to state that the ----- Co. is opposed to voltages above 500.

Voltages very much higher are safe enough above ground, but in coal mines we encounter moisture and sulphur. The apparatus will sweat; the moisture will evaporate and float through the electrical equipment, condense, and cause grounds in the case of high-voltage direct current. The result is the arcing over and burning out of various parts.

In permissible equipment it is difficult to handle high voltages in breaking any perceptible amount of current. Fine copper dust will wear off from the contact fingers and fall on the different parts of the insulation in the starting boxes. If hit with an electric arc in a totally enclosed compartment, a larger arc is started, which is very difficult to control.

We have a customer that has 150 -------permissible short-wall machines in use; the voltage at the power houses is maintained at 500. Complaints of breakdowns of the electrical equipment are practically negligible.

We have another customer that has four permissible shortwall machines; the voltage is about 650, and they are continually complaining about the starting apparatus grounding and burning out, and the motors breaking down. The coal is much easier to cut in the mines where the 650-V machines are used, nevertheless we have more complaint from these four machines than we do from the 150 machines that are in harder cutting. This same company also has considerable trouble with their substations.

We have installed machines in several mines that have high voltage, and in each case we have reports of burn-outs and troubles in general; and during the many years that we have built coal mining machinery, we have always found it unsafe to use machines in voltages above 500, and earnestly recommend that a ruling be made that permissible equipment must not be run on voltages higher than 500.

J. As a general proposition, we believe your department and the State mine departments have established a ruling that 500 volts is the maximum that can be safely used on exposed wires in the mines, and in quite a number of States, we believe they have ruled against the use of 500 volts on exposed trolley line.

Naturally, if a higher voltage can be used, it reduces the cost of transmission, also takes care of the loss from the drop in the lines in some of the

large old mines; but I would say, on general principles, it also increases the hazards, particularly if the higher voltages are handled on open lines. Take mines as a rule, the conditions are not very favorable, as the mines are liable to be damp in spots at least, and there is always danger in a fall from the roof, and if the transmission lines are on the main roadways there is always a danger of the men coming in contact with same; and in our opinion if the higher voltage is to be used, it would require extra protection and would require a somewhat different type of controller.

We are doing so little in the underground electric work where high voltages are used that we have not given this matter much study; therefore, hardly feel competent to make any suggestions in regard to what ruling should be made.

K. While we have not furnished 500-volt mine equipment and therefore have not experienced the difficulties mentioned by the one manufacturer, our experience would cause us to be very much concerned were we called upon to furnish such equipment where it would be operated on voltages from 600 to 700.

The complainant is justified in his contentions that mine operating conditions and the small space available for electrical apparatus made the use of the high voltages impractical and unsafe.

The fact that some of the State mine inspection departments have discouraged and forbidden the further installation of 500-volt equipment on the grounds that it is not in line with present ideas of safety is pertinent in answering this question

If they have found 500 volts to be unsafe and we feel that it is unsafe, we can be quite sure that any increase above 500 volts would be most objectionable.

L. It is our opinion that the position taken by the writer of the enclosure is well founded, and that 600 or 700 volts is too high voltage for operation of equipment designed to operate on 500 volts.

We realize that the reason for increasing the voltage is to take care of line drop from the power station to the point at which the machines are operated.

We would suggest, however, that instead of limiting the voltage at the machine terminals to 500 volts it would be more reasonable to require that the voltage should be not more than 10 per cent above the normal rated voltage of machines, namely: Equipment approved for 500 volts should be limited to voltage not exceeding 550 at the machine terminals.

This company is just going into permissible motor equipment, and at this time we have equipment designed for 230 volts, which is ready for test. We are interested, of course, in any regulations that may be made regarding same.

M. We do not consider ourselves competent to discuss this subject for the reason that we do not make mining machines and do not contemplate building such equipment.

In the past we have built a very limited number of main-haulage locomotives of the trolley type which operate on 500 volts, and have had no particular difficulty with these equipments, but in most States it is contrary to the statute to use such high voltage and very few mines are thus equipped.

Not being manufacturers of mining machines we can not appreciate the difficulties involved in providing the proper starting equipment adequate for higher voltages, but the control equipment customarily used with 500-volt trolley locomotives is adequate to handle the motor equipment on 600 volts or on peak currents at 750 volts ordinarily.

We regret that we can not give you any more definite information for the reason stated.

N. I most certainly am pleased to go on record that no voltage should exceed 500 volts for any use whatsoever around a coal mine. This applies to either above the ground or below the ground, but it is specially true for inside the mine.

It is time that the Government put a stop to this greatly increasing the voltage used underground, because if they don't it won't be long before we will be up to 750 volts direct current.

If I had my personal way I would not recommened even 500 volts inside a coal mine; 500 volts is all right on a street railway system, but anything above 500 volts is in no way, shape, or manner very free from hazards in a coal mine.

O. The objection made by a manufacturer of mining equipment relative to the use of 600 and 700 volts on 500-volt apparatus, should in my opinion be sustained.

The electrical apparatus such as starting box, fuse boxes, etc., designed for 500-volt machines, is always designed in accordance with certain limitations as to the creepage distance of the electrical contacts to ground. If the voltage is raised to 600 or 700 volts the margin of safety is thereby very greatly reduced, and especially when this apparatus is used in underground workings subject to corrosion, dirt, and moisture the safety factor is not as great with this piece of equipment as it would be for surface work.

I might mention an additional point which the objector did not give; namely, the commutation of a 500-volt D.C. motor is not nearly so good on 600 and 700 volts as it is on 500 volts. Even under ordinary conditions the commutator on such a motor should be redesigned for the higher-voltage operation. While this in itself might not constitute an objection as far as the permissibility goes, it certainly does hasten the time when the commutators must be inspected and attended to.

P. We have made an investigation of machines that we have in service operating on voltages above 500 and find that about seven years ago we modified our designs to make them satisfactory for voltages as high as 650. These machines have been operating in a satisfactory manner on this voltage and we can not see that there is any particular difficulty in building machines for 650 volts, although we know that the ordinary 500-volt machine will usually give trouble on high voltages. For this reason we can not agree with the manufacturer, a copy of whose letter was enclosed with yours of December 19, that the space limitations in underground machines make the use of high voltages impractical.

In this connection it might be interesting to you to know that at the present time we have on our shop floor a loading machine for use underground that will be operated on 2,200 volts.

The question as to whether high voltages are safe we believe should be determined by the mine operator and the State or Federal mining departments. We do not believe that the manufacturer should express any opinion regarding the use of high voltages - except as it might affect the machine - but that for whatever voltage may be chosen by the operator and accepted by the Government mining department the machine manufacturers should design the machines, and we have found it possible to design satisfactory machines for use on voltages of from 600 to 700.

Q. It is the writer's opinion that it is not a practical provision to add the 500-volt clause to the name-plate of permissible machines.

The voltage designations of 250 and 500 volts are necessarily only approximate and the over-compounding of the generator units to give these voltages at the machines must necessarily cause an impressed voltage on the lines of 10 to 25 per cent of the rated voltage.

There is undoubtedly much more liability to grounds and short circuits at 700 volts than at 500 volts, but in the writer's experience it is only when power has been purchased and taken directly from an interurban trolley line that voltages exceed 600 volts at the switchboard.

R. I consider the use of 500 volts underground more hazardous than 250 volts, and increasing the voltage to 600 or 700 volts more hazardous.

The manufacturers are finding it a difficult task to provide the proper protection required for permissibility with 500-volt equipment. To take care of 700 volts will be a much greater task, and I don't see the necessity for it.

Comments from Operator Group

A. Technical Paper 138 allows use of 650 volts maximum for medium voltage. This same voltage has been approved by Mining Congress Standardization Committees and Technical Paper 138 revised.

Our voltage at mines is frequently 600, and we are experiencing no trouble.

In the purchase of 500-volt equipment, mine operators should indicate maximum voltage to which apparatus may be subjected.

The objection should not make any change in the rules, which apparently have been satisfactory for a number of years.

B. Referring to your letter of December 19, 1925, relative to voltage of circuits furnishing power to permissible machines; my own personal opinion is that the limit should be placed at 500 volts on all portable machines, as these are always given the most severe usage. Stationary machinery, such as pump installation, are usually set up in dry, well-ventilated rooms, and there is not much danger of failure due to moisture or rough handling.

C. Anticipating as we did quite a number of years ago that the first installation cost of 250-volt equipment was somewhat higher and yet realizing fully the safety detail, we have in every case, with the exception of one small operation, adhered strictly to this standard. At the one small operation, where we have 550 volts, we are within a few months going to finish up entirely our available acreage, and I might say at this point we have our own 550-volt motor generator set; thus it is impossible to reach anything like the over-voltage of 600 or 700, which, most assuredly, is a dangerous practice.

D. Personally I do not approve of the use of a 500-volt electrical system for haulage and room equipment.

Where 500-volt systems are in use, it would appear best practice to hold the voltage as near 500 as is practicable.

I believe 10 per cent plus or minus would not be objectionable as all apparatus has an insulation factor of safety that will take care of 10 per cent.

A voltage of 700 is beyond the range of safe operation for 500-volt apparatus.

E. Our opinion is that the boosting of voltage as indicated is impracticable and an unsafe practice, and therefore would not recommend it for underground installations.

F. It would seem that the question as to whether or not any material hazards would be involved if the voltage of circuits furnishing power to 500-volt equipment was increased to 600 or 700 volts, would depend entirely upon the design of the permissible equipment.

On the other hand, I believe it is quite the usual procedure, to operate, under normal conditions, a 500-volt circuit at voltages considerably in excess of

this figure. This is a practice which has been brought about by the fact that mining circuits in general are of a more or less temporary nature, and for this reason it is considered good engineering to permit considerable drop in voltage under the maximum load demands.

It would therefore seem that, due to these conditions, the designer of permissible equipment should have this feature in mind and design his equipment to successfully meet the average mining conditions rather than to take the position that mining operations must be penalized in order to conform to the design of permissible equipment which would only be suitable when operating at a voltage not in excess of 500 volts.

G. You will remember that in 1912 the subcommittee of the commission to revise and codify the present Anthracite Mine Laws of -----, submitted for the consideration of the said commission a draft of an act of assembly revising and codifying the Anthracite Mine Laws of ----- which distinctly specified in the electrical section of the draft in question that the class of equipment which you have under consideration can not be operated at a potential exceeding 650 volts.

For some time back, there has been more or less agitation to limit the voltage of mine locomotives and mining machine equipment, and we have therefore only made installations for electric locomotives and other machinery at potentials which would not exceed 300 volts.

We are of the opinion that any ruling which you should make in reference to mining machines should be such as to limit them to a lower voltage than 500 rather than going higher than this amount.

H. Referring to your letter of the 19 instant regarding voltage being higher than 500 volts when furnished to permissible machines, would advise that it is our practice to operate our generating plants at a voltage of 550 to 575 volts maximum, this being done to allow a drop of from 50 to 75 volts at the place where machines are used. Of course, you will understand that the drop is based upon a considerable load, which would be higher than that used by an ordinary permissible machine; consequently, there might at times be as high as 550 volts on the permissible equipment. However, we would not recommend that the equipment be made suitable for higher voltages than 600 volts, but we think all permissible machines should be capable of withstanding and breaking a 600-volt circuit.

I. From the operator's standpoint, we are absolutely opposed to the use of voltages higher than five hundred (500) at the machinery terminals underground. We are convinced that the economy effected with voltages higher than the above is not sufficient to offset the danger to life and property by its use.

J. I do not know of any installation where the voltage has been boosted over 600 volts.

The bituminous mining law permits the use of voltages up to 650 volts in mines, which is termed "medium voltage." It is against the mining law to supply current at a higher voltage than this for any portable machines.

In making a ruling to hold a voltage down to 500 volts in my opinion would not meet with the approval of operating companies. In placing a limit on the voltages for permissible machines, I would not recommend a voltage lower than 550. It is common practice in this section to generate and supply current at the generating station with a terminal voltage of from 575 to 600. This permits a transmission loss of about 50 volts to the working face. However, it is a rare occasion that the voltage would exceed 550 volts across the machine terminals. This, I would consider the minimum for economic operation if a ruling was enforced to supply current to permissible machines.

It would be my suggestion that the manufacturers design and build their equipment to withstand voltages ranging up to and including 550 volts for permissible machines.

K. I feel that the advantages accruing from the general use of 600 or 700 volts on underground machinery are so few as compared to the hazards involved that it is indiscreet to advocate such a course.

There have been in the past two generally recognized voltages for underground mining service, viz: 250 and 500, and there does not, to my mind, exist any need for a higher voltage in underground work, which would cause more confusion to the operator, increase the hazard to property and workmen, and force the manufacturer to design practically a new line of equipment for the higher voltage.

The arcing distance to ground must be materially increased with the use of 600 or 700 volts, and due to the limited space available in mining machinery, it would become a serious problem to the manufacturer to build, and just as difficult a problem for the mine operator to maintain equipment for this voltage, especially on portable machinery.

The fire hazard in a mine would be considerably increased due to failure of the insulation on the machinery and the failure of the insulators carrying the circuits through the mine.

Even if permissible machines are used, the hazards are not all eliminated, since it would still be necessary to carry the circuits for the high voltage. Furthermore, while the machines may be maintained as permissible for some time after they are placed in operation, it becomes increasingly difficult to keep them permissible as they get older, and the high voltage becomes more and more a distinct hazard to the person or persons operating the machine.

With the practical and successful application of the automatic substation to mining service, there no longer exists any reason why a satisfactory operating pressure can not be maintained at the working face with present accepted voltages of 250 and 500.

L. I note it has been suggested that a ruling be made that in cases of approved machines the voltage at the terminal of the machine must not exceed 500 volts, because of the fact that they are limited as to space for insulating purposes.

The so-called 500-volt standard generators are really not 500-volt machines, as at no load they generate 550 volts and compound up to approximately 575 volts. This is standard practice in building generators. This 500-volt specification perhaps originated due to the fact that very few mines have sufficient copper installed, and at the face during working hours we very frequently find 500 volts instead of the generated voltage at the plant, and I believe that some manufacturers have gone so far as to design their machines for 500 volts only, excepting or encouraging, as it were, sufficient line loss to bring the voltage down to 500.

It requires very little more insulation for 575-volt operation than it would for 500 volts, and rather than encourage line loss due to insufficient copper through stamping on the name plate that no more than 500 volts shall be supplied to the terminals of cutting machines, I would suggest that it be stamped 575 volts, as this will bring us within the voltage rango of standard generators.

I might also add that if the 500-volt reading were adopted, it would mean that during off-peak hours the voltage at the face of many mines would be 575 volts, and during on-peak hours it may run as low as 400. I know of no mine where they have 700 volts at the face unless they are purchasing direct current from street railway companies, as all their generators are special, and designed for 700 volts.

M. I beg to state that I quite agree with the manufacturer. If the machine is built for 500 volts at the terminals, it is rather unreasonable to expect to work the machine under a voltage boosted up to 600 or 700 volts and expect the starting boxes, resistances, and fuse boxes built for 500 volts to stand up under this high voltage.

It is the usual custom to generate current at a higher voltage at the power station in order to take care of the line drop between the power house and the mining machine, but this must be done within reason, and at any rate the starting apparatus at the machine must be built to take care of a temporary higher voltage than the machine is expected to work under, as the operating machine will immediately take from the line momentarily the full voltage of the line, and any regulation covering the voltage at the machine must take into consideration this temporary high-line voltage in getting the machine under way.

Our people, experienced in the handling of such apparatus, strenuously object to fixing the maximum voltage on approval plates at 500 volts.

If a maximum is to be indicated on approval plates, it is suggested that an allowance of 20 per cent for over-voltage should be made, - making the maximum 300 volts for 250-volt apparatus and 600 volts for 500-volt apparatus.

N. The voltages used in our mines for coal-cutting machines and locomotives is 250, and this voltage is not under any circumstances boosted more than 10 or 15 per cent; this is done to take care of some of the voltage drop which is greater where direct current is used, as in our instance.

My view with regard to boosting voltage to the machines in general is that the voltage should never be boosted more than 10 per cent above the rated voltage of the machine, this should not be done where there is not an appreciable drop in voltage due to line loss, and the proper way to take care of line loss is to install larger feeder copper.

O. I wish to say that I am strongly in accord with anything that would tend to prevent excessive voltage on 500-volt equipment. Twenty-five years experience with this class of apparatus has convinced me that not only is the hazard and chance of personal injury increased by the use of 500-volt portable machines, but the cost of maintenance is increased as well. I am of the opinion that the savings in investment and possible decreased losses in transmission is more than offset by a higher maintenance cost.

Here in -----, additional installations of 500-volt underground mine equipment have been prohibited by the State mining law for several years. I consider it a wise move for engineers and manufacturers of electric mining machinery to exercise their influence to prevent not only increased voltage on 500-volt systems, but as much as possible to prohibit new installations of systems carrying more than 300 volts for portable mining equipment.

P. Wish to advise that when electric power was first installed in our coal mines in 1902, individual power plants were installed at each mine producing 500-volt current, the center of electric power in the mines being approximately 1 mile from the generating station. The cost of feeder lines at that time was the determining factor in favor of 500 volts against 250 volts.

In 1908, after 6 years experience with 500-volt systems, it was decided to use 250 volts in all new mines, and this has been the standard since that time.

In one of our mines, we are now changing from 500-volt equipment to 250-volt equipment on account of the difficulties mentioned in your letter, and as a matter of safety.

Under modern mining practice, the ventilating current of air is nearly saturated, which affects heavy insulated feeder cables.

We do not consider it safe to use higher than 250 volts in our coal mines.

Q. It appears as though the mining people who are running their voltage up to 600 or 700 volts on a nominal 500-volt circuit are trying to beat the conomic law of distribution of power, possibly at the same time violating State laws. Although I do not have a copy of all State laws at hand, it is my understanding that practically all of the States limit the voltage which may be carried underground. In this state, for example, all exposed wires are limited to 275 volts.

In regard to the practicability of manufacturing and maintaining permissible equipment on 700-volt circuits, I can readily see that the problem would be much more difficult on 700 volts than on 500 volts. Hence, it appears to me that the bureau would be justified in qualifying the endorsement on equipment to proper rated voltage.

R. We are unable to supply any information in reply to your inquiry of January 28, since our experience has been limited to mine voltages of from 250 to 300 volts.

S. I personally am very much against any voltage in coal mines of over 275 D.C. and 220 A.C. I think that the hazard of 500 volts D.C. and 440 A.C. is too great, and to my personal knowledge, this higher voltage has caused a very high mortality. I think it is possible to inclose a 500-volt machine so that it will probably be as safe from flame as a 275-volt, but I question very much if this can be done in a limited space for a 700-volt current. From absolutely personal interest in the production of something like 75 million tons of coal during the past 25 years, I know of only one case of a man being killed by a 250-volt circuit. Around the Rock Springs mines of the Union Pacific Coal Co while I had charge of those properties we averaged nearly a man killed a year with the 500-volt circuits.

T. I am of the opinion that good practice demands that the voltage supplied to 500-volt mining machines should, under no conditions, be as high as 600 volts. It is impracticable in the design of such equipment to provide insulation for 600 or 700 volts; in fact, it is difficult enough providing equipment good for 500 volts

As to the exact limitation of voltage I don't agree to limiting to a maximum of 500 volts, because there must be some spread between average operating voltage and maximum. I think the old accepted practice of allowing 10 per cent is sound and therefore the limitation of voltage should be set at 500 volts.

U. I have your letter of January 28 and in reply wish to say that my practice has been not to use over 250 volts on locomotives and mining machines, and I therefore do not feel qualified to say what troubles would be experienced with 600 or 700 volts on equipment designed for 500 volts, though I agree with the manufacfacturer's complaint - that on account of limited space in mining machines it is desirable not to exceed the voltage for which the machine is designed, because it is sure to cause trouble with contact fingers, resistance and commutators.

For the last 15 years I have made it a practice - in larger mines - to carry 2,300 volts in steel armored cables into the mines for the operation of motor generator sets or static transformers which were so located that the secondary lines would have the minimum length, believing that this was the most economical method and prevented disturbance of power lines.

All alternating motors over 75 hp. are operated on A.C. current and, where possible, mining machines are built for A.C. current, though certain types have not been developed excepting for D.C. current, and that is what I am using in the mines of this company.

V. In the first place we are not using any permissible machines operated with voltage of 500 or above. We feel that we would have more hazard with such a voltage than we do with our 250 to 300 volts D.C. It appears to us that the question of decision between 500, 600, and 700 volts should include an investigation of the present-day standards. As we look at it, standard generators are built to compound from 550 to 600 volts, and it would therefore be possible for any piece of equipment in the mine to receive 600 volts from such a generator. For this reason our recommendation would be to place the maximum at 600 volts D.C.

GENERAL COMMENTS BY THE BUREAU OF MINES

As a result of the foregoing comments by State inspection departments, manufacturers of electrically operated mine equipment, and operators, the wording on approval plates was modified to include the following clause: "The operating voltage of 500-volt equipment must not exceed 500 volts at the motor terminals."

In addition, bureau engineers have been more critical in the inspection of 500-volt equipment for electrical clearance and have endeavored where facilities permitted, to conduct adequacy tests on control equipment which would show weakness in design.

The comments of the State inspection group show a marked preference for 250 volts as compared with 500 volts or even higher voltages. If the question was left in their hands, undoubtedly there would be no 500-volt direct current installations, and possibly nothing in excess of 220 volts alternating current.

In the manufacturers group one finds two opposing sets of opinion, depending upon the experience of the manufacturer. On one hand there is the group of manufacturers who build mining equipment only. These companies would in general welcome a low voltage and would much rather build 250-volt than 500-volt equipment. They are those who must cater to cramped designs in order to find space for electrical accessories when the specifications for the completed machine limit the height and width of the finished product.

The other group of manufacturers build chiefly for street railway and other commercial work outside of the mines. In general, the voltage on street-railway equipment is from 600 to 750 volts and manufacturers of such equipment see no difficulty in using similar equipment in mines. As a matter of fact, very little of this equipment fits into the special designs required for mining operations, and where these same companies have had to bring out special designs, they are experiencing the same difficulties with excessive voltage on mining equipment.

In the operators' group there are also two lines of argument. The operator who has already a large investment in 500-volt equipment seems to come to its defense as a matter of company policy, while those operators who are not permitted by State law to have 500-volt installations, or from choice have adopted lower voltage as their standard, are quite frank in their condemnation of 500-volt installations when viewed from the safety angle.

It would seem that the operator who has mines where there may occur dangerous accumulations of methane or coal-dust undertakes an extra burden if he attempts to operate with a 500-volt installation, for he must exercise continual vigilance to maintain even a semblance of safety.

I. C. 6557 March, 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

HAZARDS TO UNDERGROUND WORKERS FROM INFLAMMABLE SURFACE STRUCTURES NEAR MINE OPENINGS

BY

D. HARRINGTON AND M. W. VON BERNEWITZ

I.C. 6557
March, 1932.

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

HAZARDS TO UNDERGROUND WORKERS FROM INFLAMMABLE SURFACE STRUCTURES NEAR MINE OPENINGS [1]

By D. Harrington[2] and M. W. von Bernewitz[3]

PURPOSE OF THIS PUBLICATION

Sections 120 and 121 of Article XI, Fire Control, of a proposed Mine Safety Law in preparation, to be used as desired and when approved by any State, contains these provisions:

> After the adoption of these regulations, no mine operator shall erect any nonfireproof structure within 50 feet of any mine entrance or airway. Every operating mine having nonfireproof structures within 50 feet of any mine entrance or airway, except headframes necessary for hoisting, shall have such structures fireproofed or removed and replaced with fireproof structures within six months of the adoption of these regulations. Before reopening, mines idle at the time of the adoption of these regulations shall have all nonfireproof structures fireproofed, removed, or replaced with fireproof structures as required by the foregoing sentence.
>
> Upon receipt of the written statement of a mine operator that the fireproofing of existing structures is impracticable, the chief mining engineer (of the State commission) shall satisfy himself as to the accuracy of the statement. If he concurs that fireproofing is impracticable, and if at the same time he holds the opinion that adequate protection can be provided otherwise by such procedure, he shall have authority to issue a written permit for the substitution of fireproof door at or near portals of mine openings and collars of shafts. Such doors shall fit as nearly gas-tight as possible and shall be installed in such a way that the mouths of mine openings can be closed readily from outside of structures. The effectiveness of such permit may be made contingent upon the continued compliance with any additional precautions which are necessary in the chief mining engineer's opinion to protect the lives of miners.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from the U. S. Bureau of Mines Information Circular 6557."
2 - Chief engineer, safety division, U. S. Bureau of Mines.
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The reason for inclusion of precautions of this type in a mine safety law should be obvious, but to those who may not understand why, it is only necessary to mention the fact that many lives and much property have been lost when fires have been transmitted from surface structures to underground workings, and many present-day mines have conditions such that a fire in surface structures would be likely to result in loss of life and probably of property in the mine. As the efficacy of gunited timbers in shafts, concrete-lined shafts, timbers treated with zinc chloride, fire doors, escapeways, and reversible ventilation to restrict or stop fires should be well known, these precautions will be discussed briefly, if at all; this paper covers only the hazard of lighted inflammable surface material or structures transmitting fire or poisonous gases or both into mine openings, whether they be shafts, slopes, drifts, adits, or tunnels, at metal, nonmetal, and coal mines.

METAL MINES

<u>Long Wooden Shed Connecting Surface Shops with Tunnel Mouth</u>. - At a metal mine at a high altitude in Colorado, opened by a long tunnel, safety engineers of the United States Bureau of Mines noted a long shed extending from the tunnel mouth to the carpenter, machine, and electrical repair shops. The greatest hazard at this property appeared to be that of fire; if one started in the surface shops, smoke would be drawn through the connecting shed into the mine because the tunnel was the intake. On account of the layout of the workings the smoke probably would not reach the workers, yet it would be very likely to cut off escape. A second exit could have been arranged at fairly reasonable cost, as there were several connecting shafts. Numerous other precautions which could be made available at little or no expense, were lacking. This situation is by no means unusual, and it is almost a miracle that accidents from fires at mine openings are not of almost daily occurrence.

<u>Drift Mouth Houses in a Log Building Burns</u>. - A fire occurred in a prospect mine in a western State about 8:30 a.m., on January 17, 1931, and destroyed a surface building at the drift mouth, burned some timber in the drift mouth, and caused some caving at the mine opening.

There is only one opening, and this drift, which is 1,500 feet long, branches 440 feet from the surface, and another drift is driven 500 feet left of the main drift. The mouth of the drift was housed in by a log building which contained a compressor, a 220-volt generator, and a gasoline motor. Approximately 300 electric detonators were stored in this building. An 8-inch zinc-coated pipe extending to within 20 feet of the face was depended upon to provide natural ventilation. The other end of the pipe was connected to a 30-foot chimney outside of the mine. Compressed air provided most of the ventilation.

While three men were in the mine, reports like outside blasts were heard. The manager started to investigate and encountered thick smoke about 400 feet from the drift mouth. The men broke the ventilating pipe outby the junction

of the two drifts and stuffed coats in it to prevent air circulation. They then retreated several hundred feet into the longer drift and by so doing saved their lives. The building burned down, and then the men were rescued by a crew from an adjoining mine. The cause of the fire was not determined definitely, but there is no question that much inflammable material was in the surface structures which were inflammable themselves and located too close to the mine entrance. Moreover, in this case there was but one exit and there was no one in attendance at the surface while the men were working in the mine.

Wooden Snowshed Transmits Fire to Adit Opening. - At a western metal mine a fire of undetermined origin started in the surface plant 10:15 a.m. one day in July, 1930, and extended into the mine. Five men inside were asphyxiated. The surface buildings consisted of a compressor house and a blacksmith shop about 25 feet from an adit opening; these buildings were connected with the mine by a wooden snowshed. A similar snowshed connected the sluice boxes with the surface structures. An explosives magazine was situated about 75 feet from the end of a dump. The mine is opened by three adits in a gravel hillside to recover gold. The top adit, which is the present mine, is driven level for about 1,128 feet. About 765 feet from the portal a dip slant, 90 feet long, connects with the middle adit, which was the supposed escapeway; however, as is frequently the case in mines, but especially in small mines, it was impassable at the time of the fire. The cook, wife of one of the deceased men, observed the fire and gave warning to two men working in the lower adit near its portal and they escaped. The surface structures burned quickly and soon the fire began to burn the timber in the adit. Two forest rangers with the two workers who escaped and two fishermen entered the top adit and extinguished the fire in the timber near the portal at about 12:45 p.m. One forest ranger placed a wet blanket around his head and entered the relatively thick smoke and explored the top adit; he found no one and returned, rather confused. After waiting until the smoke cleared, the rescue party entered the mine and found the five men together in the lower adit. They had been traveling with the smoke and had proceeded 550 feet before they were overcome. The fire clearly shows the desirability of providing fire doors at the entrance of mines where there is a chance that fire can communicate from the outside to the inside. This was another instance of fire starting in an inflammable surface structure near the opening of a small mine, where no one was in attendance at the surface structure while men were working in the mine. Supposedly there was a second exit, but as is very frequently the case, it was impassable. If these men had known how to barricade themselves they might have saved themselves.

What a Burning Headframe did. - In 1928, fire broke out in the surface timber yard of a metal mine and burned the old wooden headframe and at the same time ruined the new steel headframe which was almost completed. The fire was probably started by a carelessly thrown match or cigarette butt and spread rapidly; there was little time to warn men underground by telephone before the wires, which were in the main shaft, were burned out. Several sets of shaft timbers below the collar were charred, and it is probable that the shaft was saved from more or less complete destruction because the

mine had a fan. By using the reversing feature, the direction of air flow in the shaft was changed from normal downcast to upcast shortly after the fire was discovered. In all probability this action not only saved the shaft from being gutted by flame, but also prevented the approximately 125 underground employees from being trapped by poisonous gases and smoke; the men escaped into an adjacent mine. This shaft had been provided with a fire door near the surface but for some reason the doors were not used, probably because they had not been kept in condition for use. There is no question that the lives of many, if not all of these underground workers were saved by two safety features, both very frequently lacking at mines and especially metal mines: (1) A mechanical fan so installed as to allow reversing direction of air flow, and (2) a second exit, whether that exit was through an adjoining property or (preferably) in the mine itself.

Head-House Fire Endangers Six Men Underground. - In November, 1928, a fire in the head house, which included the blacksmith shop and other surface equipment, as well as the gallows frame, soon was transmitted to the shaft timbers near the surface. Not long afterwards falling pieces of burning boards and timber ignited the shaft timbers lower in the mine. A surface worker in order to warn the six persons working underground heroically climbed down the ladderway, notified the foreman some hundred feet below, and then forced his way back up the shaft through the smoke. The foreman assembled the men and unable to go through the gases and smoke in the shaft, led the men to safety through some old workings whose existence were known to few, if any one other than the foreman. Here the heroic action of one man and the special knowledge of another very probably saved several lives.

COAL MINES

Burning Surface Buildings Ignite Slope Timbers. - Between 1:00 and 5:00 a.m. one day in October, 1930, a fire of unknown origin consumed the surface buildings at a coal mine in the middle west. The fire was transmitted to the slope, the only opening to the mine, and resulted in the death of three men and the subsequent death of one other man (not employed in mining) who aided in the attempted rescue of the three men trapped in the mine - two machine men and a hoistman. The operation was a small wagon mine owned cooperatively by 10 men and had a slope driven 550 feet on a pitch of 25° to intersect the coal bed with entries turned right and left near the foot of the slope. Ventilation was by a small motor-driven fan placed near the slope mouth; a wooden brattice divided the slope down to a point where an airway was driven to the left, the return being up the slope haulage way. The fire practically destroyed the brattice as well as the timber sets for about 30 feet inby the slope mouth, and the poisonous fumes killed the three men trapped because of the fact that the only opening to the mine was on fire. This is another recent case, this time at a coal mine, where a fire of unknown origin started in an inflammable surface building adjacent to a mine opening, and where no one was in attendance at the surface, the hoistman having gone into the mine, presumably to assist with the underground work.

The small mine employing fewer than 10 men is usually allowed to do essentially nothing to safeguard workers, and only too frequently little or nothing is done by the small mine to insure safety. In some States the inspection force is given no jurisdiction over mines unless more than a certain minimum number such as 5, 6, or 10 persons are employed; few if any States have a sufficient number of inspectors to allow giving adequate attention to the safety of all of the mines, including the small ones, and in many instances when the inspector tries to enforce reasonably safety regulations, he is confronted with poverty pleas or allegations of favoritism for the large operator, and so forth.

Fan Forces Flame from a Burning Fan Housing into Air Shaft. - At a coal mine in a mid-western State fire was discovered at the air shaft about midnight one night in 1928. The main force fan was at this air shaft, and there was a reserve fan at the main shaft. That night the blades of the fan were heard striking against the sides of the casing, probably because of the overheating and burning out of bearings, with consequent throwing of the fan shafting out of line; flame soon was seen issuing from the fan house, which was constructed of inflammable material. The timbering in the air shaft burned out for a distance of about 40 feet from the surface and burning embers falling down the shaft ignited the timber and coal bed at the foot of the shaft so that the mine workings were filled with poisonous smoke and gases. Seals were built at the bottom of the two other shafts. Thirteen days later the debris at the foot of the shaft was cleaned out and work continued until the next night, when the fire rekindled; the shaft was again sealed. After 45 days the seals were opened and mine recovery started. This fire shows the hazard of not enclosing the fan and fan approach in a fireproof structure and indicates the desirability of lining shafts with fireproof material. Fortunately, nobody except the company was injured by this fire; the property loss was heavy, the cost being many times what a reasonably safe fan installation would have cost in the first place.

Another Fan Forces Flame into a Mine. - At another coal mine a wooden conduit on the surface leading from a fan at the outcrop of a lower bed to workings (58 feet apart vertically) in a higher coal bed, caught fire and the fire spread into the mine. The property is near the summit of a mountain 1,360 feet high, and the topography is rugged and irregular. A 44-inch force fan worked inside a frame building near the mouth of the air course of the lower mine, and a frame conduit, 6 by 6 by 115 feet, extended up the hillside to the mouth of the air course of mine 1, which was in the upper coal. On the night of the fire, five men were working in the upper mine, and at 11:50 p.m. the fan house was discovered to be on fire. Efforts to stop the fan were futile; it continued to blow and carry flames up the wooden conduit, which burned fiercely, and smoke was forced into the mine. Although the fan motor and belt eventually burned, the updraft continued to force smoke and flame into the air course, which was burned for a distance of 150 feet. The attempts at rescue and recovery were extremely difficult; they included hauling around the mountain and installing an electric fan; surveying in the darkness with the aid of lamps and driving a 27-foot manway from the coal outcrop to the face of an entry;

ventilation of the workings; discovery and recovery of the bodies; hauling an electric pump up the mountain and erecting it; fighting the fire with water from another mine; and finally extinguishing the fire on the seventh day.

CONCLUSIONS

A few pertinent conclusions can be drawn from the foregoing examples of surface fires which transmitted fire or smoke to mine openings.

1. To cause heavy loss of life in a mine it is not necessary for a fire which starts on the surface to ignite the whole shaft, slope, drift, or adit; smoke drawn into the workings may be enough to asphyxiate everybody, or the fire may burn only a few sets of timber at the entrance and so produce large volumes of fumes which are drawn inside. Not only should openings to all mines be fireproof, at least for some distance from the surface, but there also should be a suitable fire door at the entrance of shaft, inclined, or adit, so placed that, if necessary, not only fire but also fumes of any kind can either be kept out of the mine or sealed and retained in the mine.

2. No inflammable structure should be allowed closer than within 50 feet of any opening of any mine, and no inflammable materials such as oil, gasoline, explosives, etc., should be stored or stand, even temporarily, within 50 feet of any mine opening, regardless of whether the surrounding structures are of fireproof material.

3. Covered passageways from change houses or other surface structures to shafts or drifts are desirable in a cold or snowy climate, but such sheds should be of fire-resistant material such as light steel and zinc-coated corrugated iron, or rough lumber well gunited, and even then there should be doors so placed that fire or other gases from the surface structure can be kept from going into the mine. Some mines have a drift at shallow depth connecting shafts with change houses so the underground men need not be exposed to surface temperatures.

4. Every mine should have at least two openings which would act as escapeways if fire should start at the collar or mouth of one of them. These possible ways of escape should be kept at all times so that men in ordinary physical condition can travel through them without undue exertion or delay. In other words, they should at all times be maintained as escapeways in fact as well as in name.

5. The surface entrance to shafts or drifts should be safeguarded in various ways; if timbered, they should be gunited, or the timber before being placed should be treated with a fire-retarding preservative, or steel sets should be installed as fire breaks, or some timber sets should be removed and the rock or ground gunited, if it will stand, or preferably concreting should be done; this fireproofing should extend into the mine at least 50 feet from the surface and, if feasible, much farther.

6. If a mine has mechanical ventilation (and all mines, coal or metal, should have mechanical ventilation), it should be so arranged that the fan can be stopped easily or the air current quickly reversed if, after due consideration is given the existing conditions, it is decided to be the correct procedure to do either of these things.

7. Fire may be caused by carelessness, which includes the lack of safeguarding from ignitions by electricity, lighting, smoking, open fire, and by many unforeseen means. But provision should be made at every mine, large or small, to meet an outbreak of fire from any source.

8. Fire-fighting equipment and men who understand how to use it should be distributed so as to be available at suitable and strategic points on the surface. The equipment should include a water service of plugs, hose, nozzles, buckets, and barrels, fire extinguishers, and sand. Certain men should be trained in fire-fighting methods and given regular drill.

9. Surface fire hazards at or near mine openings should be kept constantly in mind and available precautions taken as soon as such dangers are realized. The small mine is likely to be the worst offender in allowing dangerous conditions to exist near mine openings, but the larger mines, especially metal mines, are by no means free of surface fire hazards.

The foregoing information has been compiled from reports submitted during a period of years by engineers of the United States Bureau of Mines.

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

THE IMPORTANCE OF DISCIPLINE IN MINE SAFETY

BY

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

THE IMPORTANCE OF DISCIPLINE IN MINE SAFETY[1]

By D. Harrington[2]

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We Americans fondly believe that ours is the greatest, most progressive and most civilized country in the world, but the cold statistical facts reveal that we apparently have little appreciation for the lives of those whose efforts of various kinds give us whatever greatness we have. The latest available statistics indicate that the annual death rate per 100,000 persons from accidents is nearly 80, or far higher than that of any other country in the world; it is more than double the rate in Denmark, Belgium, Netherlands, France, Sweden, Italy, Ireland, Germany, Norway, and Austria; it is practically double the rate in England and Wales and in Japan, and is from 25 to 60 per cent higher than the corresponding rate in Scotland, New Zealand, Australia, Switzerland, and Canada.

And added to this is the fact that of the main industries of the United States mining has by far the worst accident record, viewed from either the frequency or the severity basis. The National Safety Council's latest figures for 27 of the country's main industries give mining the highest or worst accident frequency rate, 74.43 in 1929; meat packing comes next with 55.94, and construction third with 50.41. In accident severity in 1929 mining also was the worst offender, with a rate of 9.99; quarrying is second with 6.11, and construction third with 4.62.

Metal mining's frequency rate in 1929 was 52.16, as against 69.25 for bituminous coal mining, 99.68 for anthracite mining, and 74.43 for the entire mining industry; accident severity for metal mining in 1929 was 5.99, as against 11.69 for bituminous coal mining, 10.87 for anthracite mining, and 9.99 for the mining industry as a whole. Although this figure indicated that metal mining is on a much safer plane than either bituminous or anthracite coal mining, the frequency rate of 52.16 for metal mining was higher in 1929 than the frequency rate of all but one of the other 26 industries about which data are available; the metal mining severity rate of 5.99, while much better

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6558."

2 - Chief engineer, safety division, U. S. Bureau of Mines.

than that of coal mining (bituminous or anthracite), was higher than all but one of the other 26 industries about which accident data are available.

Metal mining's safety record in the United States, in addition to being by no means as bad as that of coal mining, has the encouraging feature that accident rates, especially as to fatalities in metal mining, are being steadily and very definitely reduced as the years go past, though unfortunately this is not true of coal mining. The fatality rate in metal mining per thousand 300-day workers was 4.01 for the 5 years 1911-1915, 3.70 for the 5 years 1916-1920, 3.23 for the 5 years 1921-1925, 3.47 for 1926, 3.10 for 1927, 2.50 for 1928, and 3.03 for 1929. This shows a reduction of about $37\frac{1}{2}$ per cent in fatalities from the 1911-1915 rate to that of 1928, and of about 26 per cent from the 1911-1915 rate to that of 1929. The rates of similar nature and periods involving nonfatal accidents in metal mining are not so favorable, the metal mining rate of nonfatal accidents per thousand 300-day workers being 198.22 for the 5-year period 1911-1915, and 245.01 for 1926, 221.54 for 1927, 205.61 for 1928, and 200.11 for 1929; probably some of the increase in the nonfatal rate in metal mining is due to better and more complete methods of keeping and forwarding accident records during the later years.

With regard to metal mining in the Lake Superior region, the ores yield copper and iron. Copper mining is restricted to Michigan, which produces 7,000,000 to 8,000,000 tons of ore yearly. The workings are deep - some of them more than 5,000 feet. Iron mining is done in Michigan, Minnesota, and Wisconsin, which produced 16,800,000, 46,500,000, and 1,800,000 tons, respectively, in 1929. The workings are underground and open-pit in Michigan and Minnesota, and underground only in Wisconsin.

In 1929 Michigan employed 4,426 men in its copper mines and 2,364 on the surface, a total of 6,790. Of these, 16 were killed underground and two on the surface, while 1,804 were injured inside and 251 outside. At its iron mines 8,835 were employed, 5,874 underground, 2,572 on the surface, and 389 in open pits. Of these, 19 were killed underground; 393 were injured underground, 49 on the surface, and 8 in open pits.

Minnesota employed 3,926 underground, 1,725 on the surface, and 4,855 in open pits, a total of 10,506. Of these, 11 were killed inside, two on the surface, and 16 in open pits; and 374, 57, and 271 were injured, respectively, in the same workings.

Wisconsin had 682 underground and 331 on the surface, a total of 1,013. Of these, two were killed and 23 injured inside, and two injured on the surface.

As to the period 1911-1929, Michigan metal mines killed 1,399 and injured 10,919; Minnesota, 891 and 52,481; Wisconsin 139 and 9,061. The total for the Lake Superior region then is 2,429 killed and 162,461 injured in 19 years; but there has been a large and encouraging decrease in recent years in both fatalities and injuries. One fact shown by these figures is the relative safety of open-pit operations.

There is no question that present-day thought is veering strongly toward the prevention of accidents in all walks of life, including that of industry; and the more the subject of safety is carefully considered and analyzed, the more definite is it established that safe methods, safe practices, safe equipment, etc., are advantageous to employer, employee, and the general public--not only from the altruistic point of view, but also from the more cold-blooded but much more practical one of dollars and cents. Compensation laws are becoming more and yet more liberal to the injured or his dependents. This is as it should be, as there is no question that life and limb are about the most valued possessions of human beings. And there is absolutely no doubt now that the financial or pecuniary cost of industrial accidents is heavy and that it must be borne at least temporarily by the industry with, in many cases, the ultimate placing of the burden upon the consumer. In some forms of industry--and some kinds of mining may be included in this--it is difficult, in fact almost impossible, to pass this burden from the industry to the consumer; in that case about the only protection afforded the industry is that of prevention of accidents, and this will, of course, at least in large part, take care of the now important item of compensation of various kinds for industrial injuries. That cost of accidents is of importance in mining may be inferred from the present thought that the direct and indirect cost of accidents constitutes 10 to 15 per cent of the total cost to the mine operator of putting coal on the railroad cars at the mine; similar figures for metal mining are not available. Inasmuch as expert opinion on safety in industry now almost unanimously agrees that industrial safety is almost wholly a function of the attitude of the industrial operating officials, it is apparent that due to failure of those in charge of the coal mining to do their duty in the curbing of accidents, the industry is suffering an unnecessary tax to the amount of 10 or more per cent of the cost of production; and there is no question that this condition applies, possibly to a somewhat less extent, to metal mining. Since there are relatively few mines, coal or metal, in the United States now making a profit of 10 or more per cent--though many are about "breaking even," and many others are losing as little as 5 per cent--it is apparent that curtailment in the occurrence, hence also in the cost of accidents, may well in many cases result in a black rather than a red balance; and when it is realized that those in charge of our mining organizations have within their own power the curtailment, in fact, almost the elimination, of accidents, with simultaneous curtailment of the cost of accidents and in some instances with resultant actual speeding of output and of efficiency as a result of prosecuting the accident prevention work, it is manifestly time for our mine operators to "wake up" to the possibilities of personally and effectively directed accident prevention work.

With the placing of much, and in many instances of practically all, of the burden of the expense of mine accidents on the shoulders of the mine operator by compensation laws, and also with the fact that the problem of protecting the life and limb of workers is now by public opinion largely if not wholly placed on each industry (and present-day thought strongly supports the justice of both of these propositions), industrial organizations are being forced into accident prevention work not only to protect the companies but also the executives, who are engaging in this work to protect themselves

personally. The functions of the industrial executives in accident prevention work are becoming numerous and complex; if they are logical in their powers of reasoning, the executives quickly see that before the company can consistently ask or expect the workers to heed the safety suggestions of the company officials, the company and its officials must themselves "put their house in order" by laying down a concrete definite safety program; for mines this includes primarily the adoption of a logical method of mining, the installation of safe mining equipment and practices, the setting forth in definite form (preferably in a printed pamphlet) the company's minimum safety requirements, and the organization of its operating officials into an accident-prevention body. After the company has its own affairs so arranged that it can go to its workers with "clean hands" in the forwarding of a safety program, the real work begins; and this is especially the case in mining.

Notwithstanding the very general opinion that the miner, and especially the old-time miner, is the most competent person to determine what should be done to protect his life and limbs, there is no question that the numerous fallacies in this more or less universal belief are largely responsible for much if not most of the accidents in our mines, as well as for the idea that mining is inherently dangerous and that so long as mining is done not only accidents but many of them must occur and in general they must be severe in their effect. The miner is usually of limited education and of equally limited "breadth of vision" as he reads little or no mining technical literature and attends few if any meetings of mining men; his knowledge is usually only what he has been able to acquire through his own far from extensive personal experience, and even the full effect of this is marred through his lack of ability to interpret the data available to him through that experience. Not only is the miner's fund of mining information by no means great, but usually the lack of breadth of knowledge of safe and efficient mining is further handicapped by his being unduly opinionated, an affliction far too prevalent among miners and one which causes them and their employers much grief, personal, physical and financial. These and numerous other more or less correlated conditions have fairly well convinced thoughtful mine safety men that while the mine operator should be held largely (almost wholly) responsible for safety conditions and results in and around mines, the operator should have a free hand to bring about that maximum safety, the best agencies he will have available being ample efficient supervision and drastic but just discipline. The supervision should be of such nature that a competent supervisory mining official should be in personal contact with every mine worker at his working place several times per day to make certain that the worker does not "take chances," probably the worst besetting sin of mine workers as a class; and the mine officials should have full power to visit disciplinary measures on workmen who are slow to carry suggestions into effect, or who refuse point blank to obey suggestions or orders in connection with safety.

Discipline is defined in a recently published dictionary as "development of the faculties by instruction and exercise"; or it is "training whether physical, mental, or moral"; or it is "training to act in accordance with established rules"; or it is "the subject matter of instruction"; and one definition is "education." All of the above are descriptive of the type of

"discipline" which should be used very generally by mine officials in trying to achieve safety in mining; and the more usual and by no means as effective definition, "submission to order and chastisement inflicted by way of correcting and training," should be used only in very exceptional instances.

The discipline that should be used when the supervisors find some dangerous condition or practice is that of education, and oddly enough the dictionary says that the word "discipline" comes from a Latin word meaning "learn" as well as from another Latin word meaning "teach"; in other words, in using discipline the mine official who uses it logically should have learned before he tries to teach. This "learning" is a prerequisite of supervisory officials if they are to get anywhere in helping the mines as well as the workers to prevent accident. The day of the hard-boiled metal-mine boss whose main, and in many cases almost only, motto was "rock in the box," whose favorite method of getting maximum quantity of rock supposedly with minimum cost was through his burly fist or through threat of the time check, and who boasted that he had no "book larnin" is long since past in practically all mining organizations which are getting good results in mine safety and which can expect to endure in the present struggle of the survival of the fittest. The successful mine official of the future who supervises mines or parts of mines and who exercises discipline is one who is sufficiently intelligent and well educated to learn not only from his own experience but to draw freely upon the ideas and experiences of others in his own community and also throughout the entire mining and industrial world; hence he must read and must assimilate at least part of what he reads, and in addition he must be able to impart his ideas and the information he has gleaned from other mining men to those under his supervision; he must act as a teacher or educator, and in this his best weapon will be patience and kindly perseverance rather than the club of threats, fists, or the time check used by the militant mine boss of the not very far distant past.

Some relatively few workers will unquestionably resist even the best efforts of the supervisor to convey to them up-to-date ideas and methods in mining; for these the old-time type of discipline must be invoked. If kindly advice or instructions in connection with safety are rejected by the worker, there should be no hesitancy in using the type of discipline to which the mine management is agreed. Some mines give a warning for first offense in the transgressing of safety rules or practice after the worker has been instructed in the right way and fails to follow it; a second offense usually causes a lay-off of about a week or even more, and a third offense, discharge. In some organizations the local supervisor does not have the right of discharge but must refer the offender to the personnel department or to some higher official, and employment may be continued under another boss; this plan works sometimes, as personalities are likely to enter into the relations between worker and supervisor and a change of boss or possibly a change of occupation or of working place brings about harmony. However, when the supervisor feels that the worker is deliberately disobeying safety rules or practice, or is so careless as to endanger his life or the lives of others, he should act promptly and firmly, as there is no question that some workers must be compelled to work safely or they will not do it at all, and it is far better to use even harsh disciplinary methods than to cause accidents due to laxness in the enforcement of safety rules or practices.

Below are some suggestions for organizing the supervisory force so as to maintain the type of discipline to which reference is made in this paper:

1. Every mine, large or small, should have a definite actively and constantly functioning safety organization, preferably with the operating head of the company or the mine taking personal lead at the meetings and with active participation of all bosses and, if possible, with active participation by all surface and underground workers or by committees from them.

One of the functions of the safety organization should be to have some person or persons designated to investigate and report on all accidents, trivial or otherwise, and in every case the cause should be determined if possible and the data from these investigations and reports used to try to prevent other accidents or repetition of those which have occurred. United States Bureau of Mines Information Circular 6066 gives suggestions as to the data to be embodied in the reports on accidents; Information Circular 6045 gives some hints as to the formation of safety organization for coal mines and much of the data are readily applicable to metal mines; also data on metal mine safety organizations are available in United States Bureau of Mines Technical Papers 229 and 452 and in Information Circular 6351.

2. Every mining company, large or small, should educate its workers, including the bosses, in safe and efficient mining practices and methods. One of the fallacies which probably more than any other has contributed to the present unsafety of mining is the almost universally prevalent idea of mining men that the miner of long experience knows how to protect himself. The cold facts are that the usual oldtime miner reads practically nothing on safety or efficiency in mining, and hence knows little or nothing but what he himself has seen or what he has learned from some equally poorly informed miner or boss who has just emerged from the ranks of the miners. United States Bureau of Mines Circular of Information 6054 gives some hints as to methods used by the Phelps Dodge Corporation in its New Mexico coal mines in educating its miners, and the data are largely applicable to metal mining, as the same company has used essentially similar methods in its metal mines with marked success in accident reduction as well as in promoting increased output and decreased cost of output. The safety division of the United States Bureau of Mines has established a list of mining people, chiefly bosses, to whom mimeographed safety literature is forwarded as issued; those who wish to receive this safety literature can do so by applying to the Safety Division, U. S. Bureau of Mines, Washington, D. C. During the fiscal year ending June 30, 1930, 61 pamphlets on various phases of safety work in the mining and allied industries were issued by the United States Bureau of Mines.

3. There should be a supervisory official or boss for about every 25 men, always on the job during the shift to oversee the work and to protect the interests of the company as well as the safety of the employees. This is not an additional cost to the company, as definite experience in many parts of the country in practically all kinds of mines shows that if the supervisory official or boss is properly trained in his work, both as to safety and efficiency, he can much more than save the amount of his wages.

Since there are at least 25 and in some cases 100 or more mine workers for every supervisory official, it is manifestly much easier to give safety education to the officials and have them transmit the information to the other workers than to try to reach the entire working force directly. Here is where intensive supervision and discipline come in - and by discipline is meant not the utilization of "hard-boiled" methods of handling men but rather the establishment of common-sense, up-to-date regulations, seeing that they are transmitted to the men and finally insisting that these regulations be obeyed and without reservation or delay. United States Bureau of Mines Information Circular 6194 has suggestions on safety as affected by supervision and discipline, and Information Circular 6211 also has information pertinent to this topic.

4. Every mine should have in printed form and preferably in a small pamphlet which may be kept in the pocket its minimum safety (and possibly efficiency) requirements, with arrangements for alterations, additions, etc., to the data in the pamphlet from time to time. These rules or regulations should be specific, should conform to the State law but should go far beyond the requirements of the State law (as no State has mine safety laws or regulations which are anything like up-to-date or adequate) and every employee should be compelled to keep thoroughly familiar with these requirements, many progressive companies causing employees to be examined from time to time as to the safety rules and regulations. Every employee (including bosses as well as other workers) is entitled to have definite information as to the minimum safety requirements of the company, and the best-known method of conveying this information and of keeping a continuing permanent record of it is by means of printed (pamphlet) rules or regulations. Hundreds of up-to-date mining companies issue rules in pamphlet form, some of these companies being the Phelps Dodge Corporation, The Cleveland Cliffs Iron Co., the Sloss Sheffield Steel & Iron Co., Tennessee Coal Iron & Railroad Co., United Verde Copper Co., Woodward Iron Co., and Alabama Fuel & Iron Co.

5. An excellent method used by some mining companies to keep employees informed about safety and other requirements or developments is that of addressing to every employee from time to time mimeographed or multigraphed letters signed by the president or general manager or other high official. In some cases these letters are sent to the home of the worker, in other cases they are delivered to him at his working place; this is far more effective than the posting of typewritten letters. Several companies using this system have had very good safety records; however, there were also other safety methods used in addition to the sending of the safety letters. Those interested in this form of safety work can secure more information about it in United States Bureau of Mines Circular of Information 6101.

6. The field force of State or county inspectors of metal mines should be available for the forwarding of an intensive and continuing safety campaign in metal mines, but as a prerequisite these men should be authorized to attend safety meetings both within and outside of the State, such as those of the National Safety Council. In addition, they should be supplied with membership in the National Safety Council or other similar organizations to have available up-to-date data on safety in mining.

7. One of the cheapest, easiest, and at the same time one of the most effective methods of bringing safety-mindedness to the mining personnel, officials as well as miners, is the giving of training in first aid to the injured to the entire personnel. This training work does not reach anything like its full effectivensss unless every person above and below ground takes it. In the past, or up to about 1926, first-aid training was given to but a relatively few persons at any one mining plant, and this system was fully as ineffective as if but 5 or 10 persons out of every 100 in underground mining were to know how to protect themselves from falling rock or from haulage, explosives, and other types of dangers. With the entire personnel kept trained in first aid, the training becomes valuable in preventing accidents in addition to helping to minimize their ill effects if they should occur.

The training of 100 per cent of the personnel of mining plants was started on a fairly large scale in 1926, the companies supplying aid of some of their employees to the United States Bureau of Mines forces to carry the training through the entire personnel, in some instances as many as 5,000 being trained at one plant. The results are now being made apparent: One mining company with about 2,500 men trained was able the first year after the work was done to complete for the first time in its history a full year without a fatality, the best previous record for any year being eight fatalities; one company having mines with the training and others without it, reports that the accident frequency of the untrained men is eight times as great as those who have had the training; other companies report this rate as 4 or 5 to 1. This work is available essentially without cost to every metal mine, and data as to methods, results, etc., are given in United States Bureau of Mines Circulars 6020 and 6217. The training can be done partly by the United States Bureau of Mines personnel, and there is no good reason why members of the State and county inspection forces and "live wires" in the mine personnel should not aid in quickly making the entire metal mining personnel of the United States trained 100 per cent in methods of giving first aid to the injured.

8. To stimulate safety in mining, safety contests between mines or departments of mines are used to advantage. Also the giving of salary or other type of bonus to bosses whose personnel achieve or excel predetermined safety accomplishments is productive of much good. United States Bureau of Mines Report of Investigations 2617 gives data on bonus systems. Some companies give recognition to individual workers who work stated periods such as 3 months, 6 months, a year or longer without lost-time accidents; one company gives a copper button for 3 months work without a lost-time accident, a silver button for 6 months, and gold button for a year without a lost-time accident. The giving of some sort of reward for safety performances is very much worth while and this is particularly the case where there are in force penalties for infraction of safety rules.

9. Every mining company, large or small, owes it to itself, to its employees, and in a large measure to the public that all prospective employees shall be given a rigid physical examination before being employed, as well as at intervals of not to exceed a year during employment. While mining can be made relatively safe, it is futile to deny that it is a strenuous calling; and persons with defective hearing, eyesight, respiration, and heart

should not be employed in mines, for they are a menace to themselves as well as to their co-workers. It is far more unjust to employ a physically defective person and probably cause him to acquire further and more serious physical deficiencies or possibly to die in a mine or to cause his co-workers to be killed or injured, than it would be to refuse him employment upon learning of the defect or even to relieve him of employment upon his becoming physically defective to such an extent that he endangers himself or others. There is no question that if more care were taken in hiring mine employees (bosses as well as the other workers) and in their instruction upon employment and afterwards, not only safety but also efficiency in mining would be materially increased.

10. Metal mines have been fortunate in having avoided many fire disasters. The word "fortunate" is used advisedly, because the avoidance has been due more to good luck than to any considerable amount of effort on the part of the mining companies, this being largely because metal mining people in general are far too generally of the opinion that mines producing gold, silver, copper, lead, iron, and similar materials have no fire hazard. As a matter of fact, there is probably not a single mine in the United States, metal or coal, which hasn't very definite fire hazards; practically all metal mines use "powder," fuse, timber, and probably other combustibles, all use open lights, and all or practically all allow smoking; many use electricity for pumping or ventilation or haulage or other purpose. Hence, there are almost universally in metal mines the very dangerous combination of combustible material and flame; and very little burning timber or burning explosive can readily produce sufficient poisonous or asphyxiating gases to kill hundreds of persons. Every metal mine should recognize the hazard of fire either in the mine or in the flimsy timber surface structures found so generally at or around the mine opening; and every mine should have a fire prevention and fire fighting plan. The subject is far too extensive to discuss in detail here, but much information about metal mine fires may be secured in United States Bureau of Mines Bulletins 75, 188, and 204, Technical Papers 251, 314, and 363, and Information Circulars 6073 and 6557.

11. The ever-increasing use of electricity in mining introduces many new and serious hazards such as the initiation of fires and accidents from contact of persons with charged electrical installations of various kinds. Utmost care should be taken in the purchasing, installation, use, and maintenance of all electrical equipment or devices in or around mines. At least as frequently as once every month a competent electrician should make a rigid inspection and a written report of the condition of all electrical wiring, machinery, etc., in or around every mine and prompt action should be taken to follow his suggestions in keeping such installations in safe condition at all times. United States Bureau of Mines Information Circulars 6037, 6046, 6052, 6068, 6098, 6100, and 6318 give data concerning safety in the use of electricity in mining.

12. Explosives accidents continue to occur far too frequently in metal mines, and there should be the greatest care in the choice of explosives used and in the storage, transportation, charging, and firing of explosives; only

such explosive and explosives appurtenances as are of the safest types should be used. In general, electrical blasting is much safer than blasting by fuse, but electrical blasting is by no means "foolproof." If at all feasible, blasting should be done only at the end of the shift, or possibly by shot firers after the general shift has left the mine. Hints on up-to-date blasting are to be found in United States Bureau of Mines Bulletins 287 and 311, Reports of Investigations 2147, 2156, 2789 and 2790, Technical Paper 400, and Information Circulars 6043, 6056, and 6189.

13. Many mining companies are now making remarkable progress in accident prevention through use of protective clothing such as hard hats, various kinds of hard-toed shoes and other foot and leg protection, goggles, safety belts, etc. In many cases difficulty has been encountered in securing the cooperation of the workers, but usually this reluctance can be overcome if the mine officials will pave the way by themselves using the recommended materials consistently. In some cases it has been found necessary to require the use of the protective clothing, but where good judgment is used, compulsion should not be necessary, except possibly for a relatively few persons.

14. Consistence on the part of the mining companies demands that before asking or requiring the employees to alter the old for the newer and safer practices, the companies first "put their house in order." This means that mines should be well ventilated, preferably by mechanical control; that well-constructed cages or man cars be supplied to transport the men into or out of the mine; that all mines have more than one exit, and that all exits have well-placed and well-constructed fire doors for use in case of necessity; that all cages have practical and workable doors which are used at all times when men are aboard; that all of the openings to shafts, winzes, raises, or steep inclines have adequate doors or other device to prevent falling of men or material into them; that ladderways in shafts, raises, or winzes over 70° pitch have platforms at intervals of at least every 30 feet, and that ladders and ladderways be kept clean and in good repair; that all machinery is kept adequately guarded; that timber supply is ample, of correct size, length, etc., and placed reasonably close to the point of use.

There is now no question that mines can be operated essentially as safely as most of the other lines of industrial endeavor, and there is good reason to believe that safe operation is also economical and efficient operation. However, to bring about this safety will require the expenditure of much personal thought, time, and effort, not only by those who actually go into the mines to do the various kinds of work but also on the part of those who control the mine policies. Here is another case where "in union there is strength," and if the right kind of cooperation is kept in effect, it is certain that the desired results will also be forthcoming in lowered accident rate with correspondingly decreased compensation and kindred costs and also in decreased suffering to the minersand their families. In the accomplishing of these very desirable results there is no question that intensive supervision and well-directed discipline are dominant factors; and the type of discipline most likely to produce results of the right kind is that by which the supervisor conveys in one manner or other to the workers the correct, safe, and efficient manner of doing mine work.

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September, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

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GEOPHYSICAL ABSTRACTS

No. XX1X

BY

Frederick W. Lee

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]
No. 29
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1. Patents (Gravitational and Magnetic methods)

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6559."

2 Senior physicist, U. S. Bureau of Mines.

This information circular contains abstracts, by W. Ayvazoglou, from original German patents concerning gravimetrical and magnetic methods of geophysical prospecting.

Copies of the patents listed in this circular may be procured from Deutsches Reich, Reichspatentamt, Berlin, Germany.

1. GRAVITATIONAL METHODS

(352) EÖTVÖSSCHE DREHWAAGE
(Eötvös' Torsion Balance)

Werkstätten für Präcisions - Mechanik und Optik
Carl Bamberg, of Berlin-Friedenau.

343,620 Patent issued November 5, 1921.

The invention disclosed in this patent concerns the recording apparatus which is secured on the top of the vertical protecting box. By a special arrangement the light ray reflected by the mirror suspended from the filament is conducted vertically upward by means of an additional mirror, and can be observed through a window.

Claims allowed - 3.

(353) EÖTVÖSSCHE DREHWAAGE
(Eötvös' Torsion Balance)

(Addition to patent 343,620)

Askania-Werke A.-G. vormals Centralwerkstatt-Dessau und Carl Bamberg-Friedenau, of Berlin-Friedenau.

372,133 Patent issued March 24, 1923.

This patent concerns an improved means of conducting the light ray, as described in the original patent 343,620.

According to this patent the light ray is projected first to a mirror secured as low as possible and then reflected by another mirror to the recording apparatus situated at the top of the protecting box.

Claims allowed - 2.

(354) EÖTVÖSSCHE DREHWAAGE
(Eötvös' Torsion Balance)

Askania-Werke A.-G. vormals Centralwerkstatt-Dessau und Carl Bamberg-Friedenau, of Berlin-Friedenau.

378,323 Patent issued July 10, 1923.

The invention disclosed in this patent is an arrangement by which the locking device and the trip gear mechanism serving for the rotation of the balance is connected to a braking or locking device of the driving mechanism in such a way that they both work simultaneously, thus causing the driving mechanism to stop as soon as the rotation of the instrument is locked.

Claims allowed - 2.

(355) EINRICHTUNG ZUM MESSEN VON WINKELAUSSCHLAGEN
(Arrangement for measuring angles of deflection)

Gesellschaft für praktische Geophysik m. b. H. of Freiburg i. Br.

385,787 Patent issued December 12, 1923.

An arrangement is described for measuring angles of deflection by means of a mirror and a graduation scale in geophysical-survey instruments, such as torsion balances, in which the graduation scale is fixed to the telescope above the tube; a cut-away section of the telescope tube in front of the objective makes possible the passage of the light rays, emanating from the scale through the objective to the mirror and back to the eye.

Claims allowed - 2.

(356) EINRICHTUNG ZUR RELATIVEN SCHWEREMESSUNG
(Arrangement for relative gravity measurement)

Dr. Wilhelm Hammer of Freiburg i. Br.

391,777 Patent issued March 11, 1924.

This patent discloses an arrangement in which the periods of oscillation of two or more pendulums assigned for relative gravity measurements are transmitted by radio. According to this invention, the pendulums, the periods of oscillation of which are to be compared, are connected to oscillatory circuits, the emission of waves of which during an oscillating phase of pendulums is suddenly changed; this makes possible the noting of coincidences at the places of observation.

Claims allowed - 2.

(357) DREHWAAGE
(Torsion Balance)

Askania-Werke A.-G. vormals Centralwerkstatt-Dessau und Carl Bamberg-Friedenau, of Berlin-Friedenau

394,876 Patent issued April 29, 1924.

The purpose of the invention described in this patent consists of making the time during which the oscillating beam comes to rest adjustable according to circumstances. The damping device serving for this purpose is described.

Claims allowed - 2.

(358) EÖTVÖSSCHE DREHWAAGE
(Eötvös' Torsion Balance)

Askania-Werke A.-G., vormals Centralwerkstatt-Dessau und Carl Bamberg-Friedenau, of Berlin-Friedenau.

395,397 Patent issued May 16, 1924.

The Eötvös torsion balance described in this patent is characterized by an arrangement in which the influence of changes of temperature and of radiation is removed by filling the external casings protecting the balance with soft feathers; these casings serve at the same time for protecting the balance during transportation.

Claims allowed - 4.

(359) DREHWAAGE ZUR MESSUNG DER SCHWERKRAFT
(Torsion Balance for Measuring Gravity)

Askania-Werke A.-G., vormals Centralwerkstatt-Dessau und Carl Bamberg-Friedenau, of Berlin-Friedenau.

396,181 Patent issued May 27, 1924.

This patent discloses an invention in which the beam of the balance is made of a new form so that one of the weights is disposed below and the other above the horizontal plane passing through the point of suspension. By this arrangement the height of the instrument can be reduced.

Claims allowed - 3.

(360) GRAVITATIONSWAAGE NACH ART DERJENIGEN VON EÖTVÖS
(Torsion Balance of Eotvos Type)

Gesellschaft fur praktische Geophysik m. b. H. in Freiburg i. Br.

413,284 Patent issued May 7, 1925.

The torsion balance of Eötvös type described in this patent discloses an invention by which the variability of the twisting force is eliminated by producing in the torsion balance directing forces based on magnetic, electrical, or mass attraction. By this the external influence upon the suspension fibre--for example, that of temperature--is also excluded.

Claims allowed - 2.

(361) EINRICHTUNG ZUM MESSEN KLEINER KRÄFTE
(Arrangement for Measuring Small Forces)

"Exploration" Boden-Untersuchungs und Verwertungs-G. m. b. H. of Charlottenburg.

416,319 Patent issued July 11, 1925.

The measuring of small forces by means of a balance beam fastened rigidly to a torsion filament, as described in this patent, is characterized by the following special arrangement: The beam is fixed to the upper part of the filament; the lower part of the torsion filament is secured to a rod, the ends of which are inserted into special guides allowing the rod to be moved in vertical direction only. The same rod serves for keeping the filament tight.

Owing to this arrangement the height of the whole instrument is reduced considerably.

Claims allowed - 3.

(362) EÖTVÖSSCHE DREHWAAGE
(Eotvos Torsion Balance)

"Erda" A.-G., Institut für Angewandte Geophysik of Göttingen.

418,348 Patent issued August 31, 1925.

The invention disclosed in this patent is characterized by the suspension of the beam to a unicrystal filament--for example, a tungsten wire.

Claims allowed - 1.

(363) TORSIONSAUFHÄNGUNG
(Torsion suspension)

"Erda" A.-G. of Göttingen

421,726 Patent issued November 17, 1925.

This patent discloses a torsion suspension as follows: The filament passes through a hollow cylinder made from a soft metal--for example, lead; the cylinder is inserted into a clamping screw; the suspension wire can be held fast by applying a screw.

Claims allowed - 1.

(364) DREHWAAGE
(Torsion Balance)

"Erda" A.-G. of Göttingen.

421,727 Patent issued November 17, 1925.

This patent describes an arrangement by which the damping of the oscillation of the beam is obtained by using the air; for this purpose the weights are placed between two damping plates. The position of these plates, thus the amount of damping, can be regulated by means of special guide screws.

Claims allowed - 4.

(365) DREHWAAGE ZUR FESTSTELLUNG NUTZBARER LAGERSTÄTTEN
(Torsion Balance for the Determination of Ore Deposits)

Dr.-Ing. Heinrich Quiring of Falkensee b. Spandau.

423,312 Patent issued December 24, 1925.

The torsion balance for the determination of ore deposits and other gravity disturbances under the ground, as well as above it, described in this patent, is characterized by an arrangement by which the deflections of the balance beam are established by magnetic or electrical potential differences occurring between the balance beam, suspension wires of the weights, and the weights on the one side, and the nonconductors or conductors of the form of a tube, ring, box, or plate enclosing them on the other side.

The measurements of the deflections of the beam are made by means of a plate condenser.

Claims allowed - 2.

(366) VERFAHREN FÜR SCHWEREMESSUNGEN NACH EÖTVÖS
(Method of Gravity Measurements according to Eötvös)

Dr. Karl Kilchling of Freiburg i. Br.

427,530 Patent issued April 9, 1926.

This patent discloses a method of gravity measurements according to Eötvös in which the torsion balance is very slowly rotated without interruption by means of a precision watch. During this constant mechanical rotation the ocular or photographic observation can be carried out in as many azimuth positions as desired.

Claims allowed - 1.

(367) PLATTEN KONDENSATOR ZUM BESTIMMEN VON MASSEN GRAVITATIONSWIRKUNGEN
(Plate condenser for determining the gravitational effects of masses)

Dr.-Ing. Kurt Emil Dittmann of Gelsenkirchen.

428,499 Patent issued May 14, 1926.

This patent describes a plate condenser assigned for determining the excess or deficiency of mass under the ground or inside of other bodies; the invention is characterized by making the condenser rotatable around a horizontal and a vertical axis so that it can be adjusted to any position of the angle; the direction, as well as the various strengths of the effects by which the movable condenser plate is influenced according to the laws of mass attraction, can be measured.

Claims allowed - 1.

(368) VERFAHREN ZUR VERSTÄRKUNG DER DREHWAAGENAUSSCHLÄGE DURCH RESONANZ
(Method for amplifying the deflections of the torsion balance beam by resonance)

Dr. Karl Kilchling of Freiburg i. Br.

428,500 Patent issued May 5, 1926.

The method for amplifying the deflections of the torsion balance beam by resonance, described in this patent, concerns an arrangement by which the period of time of uniformly rotating balance and the torsion oscillation of the suspension are mutually adjusted.

Claims allowed - 1.

(369) METALLDRAHTAUFHÄNGUNG FÜR MESS-INTRUMENTE INSBESONDERE FÜR DREHWAAGEN

(Metal-wire suspension for measurement instruments, especially for torsion balances)

Kate Heckmann of Freiburg i. Br.

428,824 Patent issued May 14, 1926.

This patent discloses a metal-wire suspension for measurement instruments, especially for torsion balances, characterized by providing the part of the instrument enclosing the suspension wire with a heating device which can be used for tempering the suspension wire.

Claims allowed - 1.

(370) VERFAHREN UND VORRICHTUNG ZUR FESTSTELLUNG NUTZBARER LAGERSTÄTTEN UND SONSTIGER SCHWERESTÖRUNGEN DURCH LOTABWEICHUNGEN

(Method and apparatus for locating useful deposits and determining gravity disturbances by means of plumb-line deviations)

Dr.-Ing. Heinrich Quiring of Falkensee b. Spandau.

429,034 Patent issued May 17, 1926.

The method for locating ore deposits or determining gravity disturbances above the ground and under the ground, described in this patent, discloses an arrangement by which the plumb-line deviations are determined by measuring electrical or magnetic potential differences between the plumb line and a conducting or nonconducting body surrounding the plumb line. This body may have a form of a sphere, cylinder, ring, plate, or have a convex shape. The ring or the cylinder is divided into segments, each representing a plate condenser.

Claims allowed - 3.

(371) VORRICHTUNG ZUM BEOBACHTEN DER ÄNDERUNG EINZELNER SCHWERKRAFTS-KOMPONENTEN MITTELS ELEKTRISCHER PLATTENKONDENSATOREN

(Apparatus for observing the change of single gravity components by means of electrical plato condensers)

"Exploration" Boden-Untersuchungs - und Verwertungs G. m. b. H. of Berlin.

430,643 Patent issued June 21, 1926.

The apparatus for observing the change of single gravity components by means of electrical plate condensers, described in this patent, consists of one or several condensers suspended in such a manner that they occupy a position eccentrical with regard to a vertical or horizontal axis of rotation.

Claims allowed - 1.

(372) HORIZONTALPENDELDREHWAAGE
(Horizontal pendulum torsion balance)

Seismos G. m. b. H. of Hannover

432,022 Patent issued July 22, 1926.

This patent discloses the application of the horizontal pendulums, for example, of Hengler's type, for gravity field investigations. A detailed description of the apparatus is given. In order to reduce the time of observation, two apparatuses placed opposite one another may be used.

Claims allowed - 4.

(373) VERFAHREN FÜR SCHWEREMESSUNGEN NACH EÖTVÖS
(Method of gravity measurements according to Eotvos)

(Addition to patent 427,530)

Dr. Karl Kilchling of Freiburg i. Br.

432,023 Patent issued July 28, 1926.

The method described in this patent differs from that of the original patent 427,530 by an arrangement in whih only the suspended part is rotated and the deviation of attraction is observed, with regard to the azimuth of the rotation, during the rotation.

Claims allowed - 4.

(374) VERFAHREN ZUR BESTIMMUNG DER ANZIEHUNG VON MASSEN, INSBESONDERE DER ERDANZIEHUNG

(Method for determining the attraction of masses, especially that by the earth)

"Exploration" Boden-Untersuchungs - und Verwertungs-G. m. b. H. of Berlin

432,130 Patent issued July 28, 1926.

The invention discloses the following method for determining the attraction of masses. A system of bodies of masses is enclosed in a box provided with a pressure meter. The system is subject to changes of position, if the force of attraction under the effect of whichthe masses of the system are is changed. The system is brought then to its original position by the change of the density of the medium in which the system is inclosed. The value of this variation of density can be established by means of a pressure-meter and serves as a basis for determining the change of the force of attraction.

Claims allowed - 3.

(375) VERFAHREN FÜR SCHWEREMESSUNGEN NACH EÖTVÖS
(Method of gravity measurements according to Eötvös)

(Addition to patent 427,530)

Dr. Karl Kilchling of Freiburg, i. Br.

432,185 Patent issued July 28, 1926.

This addition to the original patent 427,530 discloses an invention by which a light ray is rotated together with the torsion balance; this light ray records the deviation of the attracted suspended part with regard to the azimuth of the uniform rotation. The recording is produced in the form of a curve or the light ray is exposed by a mechanical device at fixed periods of time in such a way that it serves as a time signal by which the elements of the movement of the rotation of the suspended part are recorded.

Claims allowed - 3.

(376) VORRICHTUNG ZUR VERMEIDUNG DES KLEBENS DER GEHÄNGE BEI MESSINSTRUMENTEN, BESONDERS DREWAAGEN

(Arrangement for avoiding the adhesion of the suspension part in measuring instruments, especially torsion balances)

Dr. Johann Koenigsberger of Freiburg i. Br.

432,598 Patent issued August 7, 1926.

This patent discloses a means for preventing the suspension part of a measuring instrument from adhering to other parts with which it makes contact during its oscillation; this result is attained by making the parts which come in contact from hard oxides or sulphides on the one hand and from metals or metallic alloys on the other; owing to this arrangement the parts which come in contact, instead of adhering, repel each other.

Claims allowed - 1.

(377) TEMPERATURSCHUTZ FÜR MESSINSTRUMENTE ALLER ART
(Temperature protection of measuring instruments of all kinds)

Alfred Gawron of Berlin-Wilmersdorf.

436,592 Patent issued November 4, 1926.

The invention discloses a special method of temperature protection; the protection consists of several separated covers having a form corresponding to the parts of the torsion balance or of any other measurement instrument. The covers are made from waterproof material and are provided with openings through which the observations can be made without removing the covers.

Claims allowed - 1.

(378) GEHÄNGEKLEMMVORRICHTUNG FÜR TORSIONS INSTRUMENTE
(Locking device for the suspension parts of torsion instruments)

Dr. Johann Königsberger of Freiburg i. Br.

437,215 Patent issued November 19, 1926

This patent discloses the construction of a locking device in which the suspension wire remains stretched after the instrument is locked for transportation. By this arrangement, which consists of applying a spring producing a force almost equal to that of the weight of the mass, the change of the position of the zero point after the unloading of the instrument is prevented.

Claims allowed - 1.

(379) DREHWAAGE FÜR AUSFÜHRUNG VON SCHWEREMESSUNGEN
(Torsion balance for carrying out gravity measurements)

"Exploration" Boden-Untersuchungs- und Verwertungs-G. m. b. H. of Berlin.

440,384 Patent issued February 17, 1927.

The torsion balance proposed by the inventor is characterized by placing the rotative driving mechanism, in order to preserve a symmetrical distribution of the mass of the instrument, in a central position--for example, inside of the cylinder of the foot.

Claims allowed - 1.

(380) VERFAHREN ZUR UNTERSUCHUNG DER RÄUMLICHEN DICHTEVERTEILUNG IM ERDINNERN
(Method for investigation of spatial distribution of density inside of the earth)

Seismos G. m. b. H. of Hannover.

441,170 Patent issued March 1, 1927.

In order to obtain greater accuracy in investigating--for example, by torsion balance--of spatial distribution of density in the surroundings of galleries and workings of mines and determining the position of ore deposits the density of which differs from the surroundings, the inventor proposes that the center of gravity of the suspension part of the instrument be brought to that point of the cross section of the gallery in which the forces of attraction of the parts of the masses mutually compensate each other.

Claims allowed - 2.

(381) FADENLOSE AUFHÄNGUNG VON DREHWAAGEN, HORIZONTAL PENDELN, MAGNETOSKOPEN UND DGL

(Wireless suspension of torsion balances, horizontal pendulums, magnetoscopes, and similar instruments)

Reinhard Reeh of Nanzebach, near Dillenburg.

441,505 Patent issued March 10, 1927.

The patent describes a wireless suspension of torsion balances, etc., accomplished by means of a swimmer immersed in a liquid; the invention is characterized by such a choice of the liquid, of the inclination, and the nature of the surface parts of the swimmer, and of the walls of the receptacle at the level of the liquid, that two different menisci are formed in the circular space between the immovable receptacle and the movable swimmer.

Claims allowed - 10.

(382) MESSINSTRUMENT MIT FLÜSSIGKEITS-DÄMPFUNG
(Measurement instrument with liquid damping)

Dr. Johann Königsberger of Freiburg i. Br.

441,882 Patent issued March 16, 1927.

The liquid damping for measurement instruments described in this patent is characterized by making the suspension wire of the damping body thinner at the place in which the wire crosses the surface of the liquid; by this the effect of the surface tension is lessened considerably.

Claims allowed - 1.

(383) VERFAHREN ZUR BESEITIGUNG STÖRENDER WANDUNGSHOHLRÄUME U. DG. BEI BENUTZUNG DER DREHWAAGE UNTER TAGE

(Method for eliminating disturbing hollow spaces in walls in using the torsion balance underground)

Gesellschaft fur praktische Geophysik m. b. H. of Freiburg i, Br.

443,215 Patent issued April 23, 1927.

This patent discloses a method for eliminating hollow spaces in walls of mines owing to their harmful effect on the measurements made under ground by torsion balance. The inventor recommends filling out these hollow spaces with material the density of which corresponds to that surrounding the hollow space. Sacks filled with such material may be used.

Claims allowed - 2.

(384) VERFAHREN ZUR BESEITIGUNG STÖRENDER WANDUNGS-HOHLRÄUME UND DGL. BEI BENUTZUNG DER DREHWAAGE UNTER TAGE

(Method for eliminating disturbing hollow spaces in walls in using the torsion balance under ground)

Gesellschaft für praktische Geophysik m. b. H. of Freiburg i. Br.

443,299 Patent issued April 23, 1927.

In this addition to the original patent 443,215, concerning the filling out of hollow spaces in walls in working with torsion balance underground, the inventor states that the filling can be made not only in the form of plane surfaces, as recommended in the original patent, but also in any other form for which the gravity effect may easily be calculated, as for example, the form of a cylinder.

Claims allowed - 1.

(385) VORRICHTUNG ZUR BESTIMMUNG DER ÄNDERUNG DER VERTIKAL-KOMPONENTE DER ATTRAKTIONSKRAFT IN DER VERTIKALRICHTUNG

(Device for determining the variation of the vertical component of gravity of the force of attraction in vertical direction)

Gesellschraft für praktische Geophysik m. b. H. of Freiburg i. Br.

448,003 Patent issued August 6, 1927.

The device for measuring the deviation of the vertical component of gravity, described in this patent, consists chiefly of a beam kept in horizontal position by means of a horizontal wire; the weights are suspended to the ends of the beam at different heights. The local variation of the vertical component of gravity is manifested by the inclination of the beam and is measured by means of a mirror and a telescope. Two beams may be secured to the same wire or to two wires, the weights of the two beams being fixed at opposite heights; both beams are provided with mirrors.

Claims allowed - 3.

(386) DREHWAAGE ZUR MESSUNG DER SCHWERKRAFT
(Torsion balance for measuring gravity)

(Addition to patent 396,181)

Askania-Werke A.-G. vormals Centralwerkstatt-Dessau und Carl Bamberg -Friedenau of Berlin-Friedenau.

451,477 Patent issued October 27, 1927

The original patent 396,181 discloses an invention in which the weights are fixed to the ends of the beam, one below and the other above a horizontal

plane passing through the point of suspension of the beam. The invention described in this additional patent concerns the means by which the rotation of the beam around its own axis, thus causing the tilting of the perpendicular rods to one or the other side, is prevented.

Claims allowed - 7.

(387) DREHWAAGE
(Torsion balance)

Askania-Werke A.-G. vormals Centralwerkstatt-Dessau und Carl Bamberg-Friedenau of Berlin-Friedenau.

452,040 Patent issued November 3, 1927.

This patent discloses a combination of a torsion balance with a theodolite. The theodolite may be secured to the torsion balance either between the two casings of the two balance beams, or above the common casing in which the two beams are enclosed. The torsion balance can easily be removed and the theodolite used separately.

Claims allowed - 4.

(388) EÖTVÖSSCHE DREHWAAGE
(Eötvös' torsion balance)

(Addition to patents 343,620 and 372,133)

Askania-Werke A.-G. vormals Centralwerkstatt-Dessau und Carl Bamberg-Friedenau of Berlin-Friedenau.

453,438 Patent issued December 6, 1927.

This patent concerns an improvement of the recording apparatus described in patents 343,620 and 372,133. In order to prevent the obscurity of the reading caused by the interference of the two light rays reflected by the mirrors, a special arrangement is made by which one of the light rays is screened at the moment when the other ray is used for taking the indication.

Claims allowed - 4.

(389) VERSCHLUSS FÜR FLÜSSIGKEITS DÄMPFUNGSGEFÄSS
(Device for closing liquid-damping receptable)

Dr. Johann Königsberger of Freiburg i. Br.

453,720 Patent issued December 15, 1927.

The device for closing liquid-damping receptacles used in measuring instruments (magnetometers, torsion balances) described in this patent, consists of one or several iris diaphragms and a rubber ring through which the wire passes. By closing the diaphragm the rubber ring is compressed, thus preventing the flowing out of the liquid.

Claims allowed - 2.

(390) DREHWAAGE
(Torsion balance)

Askania-Werke A.-G. vormals Centralwerkstatt-Dessau und Carl Bamberg-Friedenau of Berlin-Friedenau.

456,039 Patent issued February 15, 1928.

The invention discloses a system of damping by means of electromagnets; the arrangement for producing the damping magnetic field is connected with the torsion balance box; a weight fixed to the horizontal arm serves as a brake copper. The copper weight has the form of a parallelepiped, the longitudinal axis of which is perpendicular to the axis of the beam.

Claims allowed - 3.

(391) GRAVITATIONS-MESSER ZUR MESSUNG DES VERTIKALEN GRADIENTEN DER ERDSCHWERE

(Gravity measuring instrument for measuring the vertical gradient of gravity)

Dr. Wilhelm Schweydar of Potsdam.

456,484 Patent issued February 25, 1928.

The invention concerns an apparatus for measuring the vertical gradient of gravity which is based on the comparison of the bifilar twisting moment of gravity with the torsion moment of a spiral spring. To eliminate the influence caused upon the length of the spiral spring by temperature changes this patent discloses an arrangement in which the spiral spring is replaced by a wire, the upper end of which is carried by a balance beam brought to the horizontal position by means of a counterweight.

Claims allowed - 3.

(392) DREHWAAGE NACH EÖTVÖS
(Torsion balance according to Eötvös)

Dr. Stefan Rybar of Budapest.

458,354 Patent issued April 4, 1928.

This patent describes a torsion balance according to Eötvös with a photographic registering device. The improvement concerns an arrangement by which the sensitive plates are secured below the measuring pendulu .

Claims allowed - 3.

(393) EINRICHTUNG ZUM EINSTELLEN DES AZIMUTS BEI DREHWAAGEN
(Device for adjusting the azimuth at the torsion balances)

Gebrüder Haff Ges. m. b. H. of Pfronten-Ried.

458,555 Patent issued April 13, 1928.

To adjust a torsion balance to various azimuths, ordinary stop screws or a ring provided with cuttings are generally used. This requires troublesome operation if, for example, five azimuths instead of three are to be taken. This disadvantage is according to this patent removed by applying two or several rings provided with cuttings which are fixed to a tube and can be displaced in vertical direction.

Claims allowed - 2.

(394) DREHWAAGE NACH EÖTVÖS
(Torsion balance according to Eötvös)

Dr. Karl Kilchling of Freiburg i. Br.

458,556 Patent issued April 13, 1928.

This patent describes an arrangement by which the torsion-balance box is suspended to a holder by a wire, thus making its special adjustment to horizontal position by means of levels unnecessary.

Claims allowed - 3.

(395) DREHWAAGE ZUM MESSEN KLEINER GRAVITATIONS KRÄFTE
(Torsion balance for measuring small gravity forces)

"Exploration" Boden-Untersuchungs-und Verwertungs-G. m. b. H.
of Berlin.

463,128 Patent issued July 24, 1928.

According to this invention one of the two weights of the balance is secured to a rigid rod directed upward, the other weight remaining as usual at the end of the balance beam. By this arrangement the total height of the instrument is considerably reduced.

Claims allowed - 2.

(396) DREHWAAGE ZUM MESSEN KLEINER GRAVITATIONS KRÄFTE
(Torsion balance for measuring small gravity forces)

(Addition to patent 463,128)

"Exploration" Boden-Untersuchungs-und Verwertungs-G. m. b. H.
of Berlin.

463,673 Patent issued July 31, 1928.

This patent discloses a further improvement of the original patent 463,128, which is characterized by lowering the center of gravity of the system by securing a special weight below the center of the balance beam. This arrangement makes it possible to reduce still more the total height of the balance.

Claims allowed - 2.

(397) DREHWAAGE FÜR GRAVIMETRISCHE UNTERSUCHUNGEN
(Torsion balance for gravimetrical investigations)

Ervand Kogbetliantz of Paris.

467,248 Patent issued November 13, 1928.

This patent discloses a torsion balance provided with three beams secured at equal angles one to another and suspended to a common wire.

Claims allowed - 3.

(398) DREHWAAGE ZUR MESSUNG VON SCHWERKRAFTS-UNTERSCHIEDEN
(Torsion balance for measuring gravity variations)

"Exploration" Boden-Untersuchungs-und Verwertungs-G. m. b. H.
of Berlin.

467,704 Patent issued October 29, 1928.

The torsion balance described in this patent is constructed so that the two horizontal bars, to which the weights are securud are connected one with another by a vertical cross arm. This cross arm is suspended to the wire directly or by means of a lateral additional piece. The connecting arm can be made from a tube and the suspension wire placed inside of the tube; the center of gravity can be lowered by fixing a special weight to the connecting arm or to the lower arm.

Claims allowed - 5.

(399) DREHWAAGE
(Torsion balance)

Askania-Werke A.-G. vormals Centralwerkstatt-Dessau und Carl Bamberg-Friedenau
of Berlin-Friedenau.

478,359 Patent issued June 24, 1929.

This patent discloses an improvement of the invention described in the original patent 456,039. The improvement concerns the annihilation of the effect of the magnetic remanence upon the brake copper after the closing of the circuit by which the results of the measurements are influenced.

Claims allowed - 4.

(400) VORRICHTUNG ZUR ANBRINGUNG VON BASISLINIEN AUF DEN PHOTOGRAMMEN WÄHREND DER PHOTOGRAPHISCHEN REGISTRIERUNG BEI DREHWAAGEN

(Device for drawing base lines on photograms obtained in torsion balances during photographic recording)

Gebrüder Haff G. m. b. H. of Pfronten-Ried.

480,443 Patent issued October 17, 1929.

The device for drawing base lines on photograms is characterized by securing a special immovable lever, provided with a sharp point. By means of light pressure the point is pressed against the photographic surface and draws on it, owing to the movement of this surface, a fine line.

Claims allowed - 1.

(401) DREHWAAGE MIT ZWEI GEHÄNGEN
(Torsion balance with two suspensions)

"Exploration" Boden-Untersuchungs und Verwertungs-G. m. b. H. of Berlin.

482,886 Patent issued September 26, 1929.

The invention described in this patent concerns the arrangement by which the two beams of the torsion balance form one with another an angle of 90° or of nearly 90°.

Claims allowed - 1.

2. MAGNETIC METHODS.

(402) ERDMAGNETISCHES VERTIKAL-INTENSITÄTS-VARIOMETER
(Earth magnetic vertical-intensity variometer)

Otto Töpfer und Sohn, of Potsdam.

152,489 Patent issued June 21, 1904.

The patent involves a vertical-intensity variometer in which the magnet system, which is placed in the inclination line, is brought to a certain, for example, normal, position by the torsion of one or several horizontal filaments or wires.

Claims allowed - 1.

(403) VORRICHTUNG ZUM SCHUTZ DER AUF TORSION BEANSPRUCHTEN ACHSEN BEI MESSINSTRUMENTEN, INSBESONDERE FÜR ERDMAGNETISCHE BEOBACHTUNGEN GEGEN SCHNELLE TEMPERATURSCHWANKUNGEN

(Arrangement for protecting axles of measuring instruments based on torsion, especially those used for earth-magnetic observations, against quick changes of temperature)

"Erda" A.-G. of Göttingen.

411,093 Patent issued March 10, 1925.

According to the invention described in this patent, the influence of quick changes of temperature upon the wire or fibre used as a torsion axis is lessened by an arrangement in which such axes are surrounded with oil or similar poor heat conducting liquid.

Claims allowed - 1.

(404) INSTRUMENT ZUR MESSUNG SÄMTLICHER BESTIMMUNGSGRÖSSEN DES ERDMAGNETISCHEN FELDES

(Instruments for measuring the values of all the components necessary for determining the earth-magnetic field)

"Erda" A.-G. of Göttingen.

415,872 Patent issued July 2, 1925.

The instrument described in this patent serves for measuring the values of all the components necessary for determining the earth-magnetic field; it is characterized by an arrangement in which a magnetic needle, rotatable around a vertical axis, and serving for measuring the horizontal intensity and declination, is fixed above or below the magnetic vertical-intensity field balance in such a manner that the distance between the magnetic needle and the beam of the field balance can be adjusted.

Claims allowed - 2.

(405) VERFAHREN ZUR ANSCHAULICHEN WIEDERGABE VON MAGNETISCHEN FELDMESSUNGEN

(Method for apparent representation of magneti field measurements)

"Erda" A.-G. of Göttingen.

421,899 Patent issued November 20, 1925.

This patent describes a method of representing by small sticks the relation to each other of vectors of disturbance calculated for each point from measured values; these small sticks, by which the established values and directions are represented, are put at those points of the map or of the plastic model of the region under investigation which correspond to the points of observation on the terrain.

Claims allowed - 1.

(406) LOKALVARIOMETER FUR MAGNETISCHE KRAFTFELDER
(Local variometers for magnetic fields)

Dr. Hans Haalck of Berlin-Steglits.

426,772 Patent issued March 18, 1926.

The localvariometer for magnetic fields disclosed in this patent is characterized by an arrangement in which two or more coils, of equal area of turn are fixed and connected at different places of a common rotation axis

in such a way that their areas of turn lie in the same plane and are directed parallel to the rotation axis, and that the alternating currents produced in them during the rotation have opposite directions. The coils are moved by means of a driving device.

Claims allowed - 2.

(407) VERFAHREN ZUR FESTSTELLUNG GERINGER LAGERVERÄNDERUNGEN VON KÖRPERN IM BEREICH VON KRAFTFELDERN

(Method for determining small changes of position of bodies within the range of fields of force)

Dr.-Ing. Heinrich Quiring of Falkensee b. Spandau.

437,365 Patent issued November 16, 1926.

Method for determining small changes of positions of bodies, for example of a plumb line, within the fields of force is characterized by the following: In order to determine the electrical, magnetic, or any other physical potential variation, or variation of oscillation caused by the change of the position of the body, the circuit which serves for measuring the change of the capacity, or of the oscillation, is closed only for a short period by utilizing the inertia of the measuring device and after this device is adjusted to the new position of equilibrium.

Claims allowed - 1.

(408) MAGNETISCHES INSTRUMENT.
(Magnetic instrument)

Ernst Ruhstrat of Göttingen.

440,899 Patent issued February 17, 1927.

This patent discloses a magnetic instrument, with two magnetic needles oscillating horizontally, for measuring the horizontal intensity in which two directing magnets are secured one above the upper needle and the other below the lower needle; the force of these magnets is calculated so that one needle is kept in a direction opposite to the earth's field and the other in the direction of the earth's field; the two needles are thus each in a weak field, of almost equal force but opposite direction, which is composed from the earth's field and directing field.

Claims allowed - 1.

(409) MAGNETISCHE WAAGE
(Magnetic balance)

Dr. Hans Haalck of Berlin-Steglitz.

441,171 Patent issued February 21, 1927.

The invention concerns a magnetic balance by which the measurements of the vertical intensity as well as of declination can be made with an accuracy equal to that of instruments generally used. The beam of the balance described in this patent consists of two or more magnetic systems crossing one another by any angle; the oscillation plane of the beam is made rotatable so that the adjustment of the balance beam to the zero position is possible.

Claims allowed - 1.

(410) VARIOMETER FÜR HORIZONTAL UND VERTIKALINTENSITÄT
(Variometer for horizontal and vertical intensity)

Ernst Ruhstrat of Göttingen

455,437 Patent issued February 1, 1928.

This invention concerns the development of the horizontal variometer with two magnetic needles. By this development the measuring of the vertical intensity is made possible also. Operating the apparatus is simplified besides.

Claims allowed - 2.

(411) MAGNETISCHES MESSINSTRUMENT
(Magnetic measuring instrument)

Ernst Ruhstrat of Göttingen

455,438 Patent issued April 5, 1928.

This patent discloses a development of the original patent 440,899 in which instead of two directing magnets only one is used. The directing magnet may be replaced by a coil through which a circuit produced by an adjustable direct current of a fixed direction and strength passes.

Claims allowed - 1.

(412) VORRICHTUNG ZUR VERKLEINERUNG DES TEMPERATUREIN-FLUSSES BEI MAGNETISCHEN WAAGEN

(Arrangement for diminishing the influence of temporature in magnetic balances)

"Exploration" Boden-Untersuchungs-u. Verwertungs-G. m. b. H. of Berlin.

471,228 Patent issued January 30, 1929.

The invention for diminishing the influence of the temperature concerns an arrangement by which the reducing of the magnetic moment in case of the increase of the temperature is compensated in a simple way. This compensation is obtained by the displacement of the center of gravity of the balance beam closer to the point of support of the balance beam.

Claims allowed - 1.

INDEX [3]

3 - The first figure refers to the number of the abstract, the second to the method of prospecting as indicated in the Table of Contents, and the third to the page.

I. C. 6560 March, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

PUMICE AND PUMICITE

BY

Paul Hatmaker

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I.C. 6560
March, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

PUMICE AND PUMICITE[1]

By Paul Hatmaker[2]

CONTENTS

INTRODUCTION

Pumice and pumicite are siliceous volcanic substances quite similar in chemical composition, but somewhat different in physical properties, manner of formation, and mode of occurrence. Production in the United States has increased from 23,271 tons valued at $94,943 in 1910, to 56,843 tons valued at $336,099 in 1930. Both pumice and pumicite are included in these figures as it has been impractical to collect statistical data on the separate commodities.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6560."

2 - Mining engineer, building materials section, U. S. Bureau of Mines.

Pumice occurs in California and Oregon, but much of the pumice consumed in the United States is imported from the island of Lipari near Sicily, either in pulverized form or as lump pumice which may be subsequently ground in this country. Pumice is used mainly as an abrasive. It is also used in construction as an insulating medium, as an aggregate in concrete and acoustic plaster, and in the manufacture of lightweight building units, the development of which is more extensive in Europe than in the United States.

Kansas and California produce most of the domestic pumicite, although relatively small quantities are mined in a few other States. Pumicite is used largely as an abrasive agent in scouring soaps and cleansing compounds, and as an admixture in concrete. It is used to some extent in various abrasive work as a substitute for pulverized pumice.

Assurance of adequate markets is an important consideration for prospective producers. Consumers usually buy abrasive material on the basis of experience as to its suitability to their own particular requirements and are seldom interested in new supplies so long as they are satisfied with that being used. Large tonnages for concrete work generally are bought on the basis of exhaustive preliminary tests on material from the most economical sources.

Pumice or pumicite from different deposits is rarely identical in physical properties. Variations in texture and composition affect the value of any particular deposit and must be fully investigated when new enterprises are being considered. Efficient abrasive material must be hard, sharp, and uniform in texture and particle size. Abrasive qualities depend to a considerable extent upon the shape of the individual particles. In many pumicite deposits the particles are flat or plate-shaped, and consequently tend to slide over a surface rather than to abrade it. This action naturally impairs the value of the material as an abrasive, but is an actual advantage in material used for cleansing and scouring compounds. The presence of sand grains, crystals of feldspar, or other foreign minerals is harmful because of the liability of a single grain to scratch polished surfaces ruinously. A like hazard exists with lump pumice having nonuniform texture or cell structure. In construction work, the chemical as well as the physical properties of the material are important.

Many deposits are erratic as to shape and extent. For this reason it is advisable to do considerable preliminary exploration work in order to make certain that a given deposit is large enough to warrant any extensive expenditures for equipment and development.

The preparation of pumice and pumicite for market appears to be simple, but producers must realize that the pumice and pumicite industry as a whole is highly specialized and that careful technical and scientific control in all departments is eminently worth while. Too little attention has been given in the past to this phase of the domestic industry, and undoubtedly this accounts, partly at least, for the present strong market position of imported material. As new uses continue to be developed, an even greater appreciation of utilization problems will be demanded from the producers.

Successful exploitation of new deposits, therefore, depends first upon markets; second, upon the ability to produce at a reasonable profit such material as will adequately satisfy specific market demands; and third, upon sufficient vision to evaluate, and courage to conform with, changing industrial conditions.

DEFINITION

Pumice is a cellular, glassy lava, usually similar in composition to a rhyolite. The term, as commonly applied, means material essentially unaltered since its formation.

Pumicite, on the other hand, is an accumulation of finely divided glass-like particles blown from volcanoes during periods of eruption and subsequently more or less classified while transported by winds. It may be wholly unconsolidated, or the individual particles may be more or less cemented together or agglomerated. Such material is also called volcanic dust, volcanic ash, tuff, and tufa. For this material the term "pumicite," however, is now commonly accepted throughout the industry, and is to be preferred to the term "volcanic ash" because such material is not the result of combustion. The use of tufa as a synonym for pumicite should be discouraged, because in correct usage tufa is a porous calcium carbonate deposited from solution.

Barbour[3] states that --

> Pumicite has been variously called geyserite, volcanic dust, volcanic ash, Gibson grit, diamond grit, native pumice, silica, and the like.*******For all of these we have substituted the name pumicite, a self-explanatory term, which we have used for some years. It is unfortunate that volcanic dust should ever have been known by the misleading term, "volcanic ash." However close the resemblance may be, it is not the product of combustion like ash, but instead is a powdered, glassy rock. In the outset it was known in Nebraska as geyserite, because supposedly produced by geysers. Survival of this conception is to be noted in Geyserite soap, which is made of pumicite. Of late pumicite has acquired the indefensible name of silica. Though a silicate, it is not silica, and should not be so called. It is natural glass, atomized by volcanic explosions. Block pumice, also natural glass, when atomized by machinery becomes the powdered pumice of commerce. Our beds of volcanic dust may be called native pumice, and the name pumicite seems appropriate and self-defining.

Tuff is a more inclusive term than pumicite and may be applied to fragmental pumice and to pumicite or other material explosively ejected from volcanoes. The pieces may be sorted or unsorted, consolidated or incoherent.

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3 - Barbour, Erwin H., Nebraska Pumicite: Nebraska Geol. Survey, vol. 4, pt. 27, 1916, p. 358.

DESCRIPTION

Appearance

Pumice looks like a glassy froth with innumerable vesicles or cavities, usually somewhat elongated and parallel one with another. In the better-grade deposits these congealed or solidified bubbles are quite uniform in size and character. Most pumice is white to gray, although certain varieties, because of impurities or a basic composition, may have other colors such as shades of red, yellow, brown, or black. Most pumice sparkles in bright sunlight but some varieties have an earthy appearance.

Pumicite, on the other hand, looks to the unaided eye like very fine closely-packed glassy sand. In the purer deposits the color is white to light or bluish gray; impurities, however, may cause a yellow, brown, or red stain. Minute sparkles are readily noted in bright sunlight. Magnified particles show marked angularity resembling powdered glass. Many are rather flat, others are triangular or dagger-shaped and some are four or five sided, still preserving, however, angular characteristics. Fluted structures parallel to the long axis are common. Most particles appear clear, but some grains are translucent and others exhibit an opaque ropy structure like fused glass.

Chemical Properties

Pumice and pumicite produced from acid magmas are similar in composition to many igneous rocks such as rhyolite and granite, and are composed mainly of silica in the form of complex silicates of aluminum, sodium, potassium, calcium, magnesium, and iron. Material of this nature is relatively inert chemically and does not effervesce in hydrochloric acid as do carbonates, a property sometimes helpful in identification. Material from some deposits may show an acid reaction with litmus, due probably to traces of sulphur dioxide.

An analysis of typical pumice and pumicite would show about the following chemical composition:

	Per cent
Silica (SiO_2)	72.0
Alumina (Al_2O_3)	14.0
Potash and soda (K_2O and Na_2O)	7.0
Lime and magnesia (CaO and MgO)	2.5
Iron oxide (Fe_2O_3 and FeO)	1.0
Loss upon ignition	3.5

Individual analyses vary somewhat from these figures and some material contains small quantities of other substances such as titanium oxide. Pumice from other than an acid magma would contain less silica and more iron oxide, lime, and other bases.

Physical Properties

For the individual particles or cell walls the hardness of pumice and pumicite is 5.5 to 6, or about as hard as feldspar and slightly softer than quartz. The true specific gravity is approximately 2.5, although the apparent specific gravity may be somewhat less. Lump pumice, because of its cellular structure, may even float readily on water, indicating a specific gravity of less than 1. The grains are brittle, but the sharpness persists even when finely pulverized. Mixed with water, pumicite settles, with no tendency to form a claylike suspension.

In texture, pumicite varies from loosely consolidated particles to flint-like material with the individual grains tightly cemented together. The size of individual fragments also varies considerably. Landes[4] lists 37 screen analyses of Kansas material, of which the finest and the coarsest are as follows:

	Rawlins County (finest) per cent	Hamilton County (coarsest) per cent
+20 mesh (0.833 mm.)	0.15	13.0
-20 +100 mesh (0.833 mm. to 0.147 mm.) .	3.0	39.0
-100 +200 mesh (0.147 mm. to 0.074 mm.) .	11.0	39.0
-200 +300 mesh (0.074 mm. to 0.046 mm.) .	28.0	7.0
-300 mesh (-0.046 mm.)	58.0	2.0
	100.15	100.0

Eardley-Wilmot[5] gives the following screen analysis of a sample from the Deadman River region, B. C.

	Mesh	Per cent
Retained on	35	0
	48	0.10
	65	0.30
	100	0.60
	150	0.80
	200	14.50
Through	200	83.60
		99.90

4 - Landes, Kenneth K., Volcanic Ash Resources of Kansas: State Geol. Survey of Kansas, Bull. 14, April 15, 1928, p. 13.

5 - Eardley-Wilmot, V. L., Abrasives. Part I, Siliceous Abrasives: Canada Dept. of Mines, Mines Branch, 1927, p. 89.

USES

Pumice

The finest grades of pulverized pumice are used for finishing silverware, watch cases, and other kinds of metal ware. Much is consumed in the electroplating industry for cleansing, scouring, and polishing. The very finest air-floated powder is used in certain dental preparations. Pulverized pumice is also used in making mechanics' soaps, for cleaning plate glass, in the manufacture of combs, pearl buttons, in pouncing paper for finishing hats, and for cutting and beveling glass.

Ground pumice has been used widely in finishing wood surfaces, but this utilization is confined mainly to the highest grades of furniture such as pianos. About 12 years ago large quantities of pumice were used for finishing automobile bodies, but this use has diminished considerably. Some pumice is also used in tumbling barrels. Ground pumice has found some utilization in putting a linen finish on celluloid collars, but this use has been abandoned.

Lump pumice is used for hand rubbing of stone, such as marble, for rubbing down paint surfaces, for finishing automobile bodies, for finishing leather, in lithographic work, and to a considerable extent in the electroplating industry for cleaning buffing wheels. A small quantity of the highest-grade material is used as a toilet article.

The following information relating to the various grades and uses of ground and lump pumice was derived mainly from trade literature of the James H. Rhodes Co. This company, however, uses different designations for its ground material; their FFFF grade is known as No. 2600, FFF as 2601, and so on to grade 10 which they describe as No. 2613.

Ground Pumice

Grade	Mesh	Utilization
FFFF	Very finest air-floated powder	Extremely fine work.
FFF	-200	Finishing auto bodies and for certain kinds of soap.
FF	-160	Do.
F	-140	Do.
0	-100	Glass cutting and silver-plate finishing.
$0\frac{1}{2}$	-80	Glass beveling and piano finishing.
$0\frac{3}{4}$	---	Do.

Ground Pumice (Continued)

Grade	Mesh	Utilization
$0\frac{1}{2}$	-60	Grades $0\frac{1}{2}$ to $1\frac{1}{2}$ are used for rough rubbing in piano factories; in the manufacture of combs, pearl buttons, and mechanics' soap.
1	-50	
$1\frac{1}{2}$	-48	
2	-40	Finishing certain kinds of pearl buttons.
3	-30	Do.
4	---	Coarse rubbing of hard materials.
7	From sand to	Used in tumbling barrels.
10	pea size	Do.

Lump Pumice

Grade	Description	Utilization
AA	Small, very finest quality.	Automobile finishing.
A	Little larger, almost same grade and quality as AA.	Do.
XXC	Size of one or two fists.	Do.
BBC	Approximately same as XXC, but inferior grade.	Do.
WW	....	Finishing stone and marble.
LIT	About 7 inches in diameter.	Lithographic work.
HP	Similar to LIT, inferior grade.	Finishing leather.
1717	Lightweight, in bags.	Electroplating work.
BD	....	Cleaning buffing wheels in electroplating industry.

Pumice has been used in the manufacture of vinegar as a porous packing material for the generators which convert dilute alcohol into acetic acid. Pumice is utilized in construction because of its light weight and its insulating properties. The manufacture of building units using pumice as aggregate has been developed much more intensely abroad than in America, although recently considerable interest has been shown in various lightweight building materials in this country. In southern Italy natural pumice is used between asphalt coatings for insulation of floors and roofs. Pumice aggregates have been used extensively in Germany in manufacturing plates, hollow blocks, and tiles for flooring. The advantages claimed for such material are lightness, heat and sound insulating qualities, incombustibility, artistic rough surface if laid up without any coating, and a surface to which plasters readily adhere if coatings are used.

Tuff has been used successfully as a building stone in Arizona and in California, and pumice stone has been utilized as a building material to

some extent in other localities. Bricks containing pumice and cement have been manufactured in the upper Rogue River district of Oregon for a number of years.

Another use for pumice in construction is in the form of acoustic plasters. Much interest is manifest at present in the relation of sound to human comfort and efficiency. The development of radio and talking pictures, the modern trend toward apartment dwellings, increase in traffic and other city noises, and other factors have stimulated research as to how sound may be controlled, both as to its reception and its insulation.

Sound reverberation is especially objectionable in theatres, moving picture stages, auditoriums, churches, and similar places. Its control is based mainly upon proper design of interiors as to shape and size, and upon the character of the wall surface. The purpose of acoustic plaster is to absorb sound waves striking the walls, with as little reflection as possible. The value of acoustic material is dependent largely upon the character of the exposed surface itself, the material just underneath having little to do with acoustical properties. Pumice has been used in place of sand for preparing acoustic plasters and is said to have given satisfactory results. It is reported to the bureau that the sizes used for this type of work are usually minus 8 to plus 30 mesh.

Sound insulation or sound deadening material is desired for places requiring a degree of soundproofness. Radio broadcasting studios, offices, apartments, hotels, and other kinds of buildings require sound insulation in varying degrees. Pumice used in plaster or other types of material has been found to possess certain sound deadening qualities. For this work a grade of minus 10 to plus 20 mesh has been used, whereas minus 20 to plus 40 mesh has been used for interior stucco, applied directly on the base plaster. A considerable field exists for further development of both acoustic and sound deadening material.

Lormand[6] quotes Bigot as saying that pumice can be pulverized and then agglomerated with "silicates" to build "a new type of furnace for glass works, the radiation of which is very slight, and which allows work to be carried out quite close to the furnace itself, thus reducing the amount of labor necessary."

Pumicite

Pumicite is used extensively as an abrasive in cleansing and scouring compounds, in mechanics' soap and preparations of similar character. Waste vegetable oils such as cottonseed oil may be saponified and used as the soap content.

As a substitute for ground pumice, some pumicites are used in electroplating with gold, silver, and nickel, and in the manufacture of pearl buttons. Other uses as an abrasive are in tooth pastes and powders, metal polishes, and rubber erasers. It can also be molded into tablet form for household and toilet use.

6 - Lormand, Charles, Paris Letter: Jour. Ind. Eng. Chem., vol. 13, Feb., 1921, pp. 171-172.

Pumicite is used to a considerable extent as an admixture to concrete in dams and other types of massive construction, in grain elevators, and also for general concrete work. It is claimed that 4 to 5 pounds per sack of cement added to the mix results in greater uniformity of the concrete, increases the workability of the mix, and hinders segregation of the other aggregates. The use of pumicite as a cement admixture in the construction of grain elevators is said to facilitate slipping the forms upward as the concrete work progresses.

In the construction of the Los Angeles Aqueduct pumicite was blended in equal parts by volume with hydraulic cement to make what was called "tufa cement." The details of this utilization are fully described in the Transactions of the American Society of Civil Engineers, vol. 76, 1913, page 520, and the results are summarized in the "Final Report, Construction of the Los Angeles Aqueduct," published by that city. Pumicite was also used in the Niobrara Dam at Valentine, Nebr. Besides having been employed in a number of other structures in the Western States, a considerable tonnage is reported under contract for the Los Angeles flood control district.

Pumicite has been used to some extent as an insulating material for packing steam and water pipes, for lagging boilers, and for lining cold-storage rooms. It has also been used in filter cells, in paints as a filler, and in sweeping compounds. Much of the pumicite produced is consumed by soap manufacturers who have their own specifications for the material, depending upon the product. To some extent the classification FFF, FF, F, O, $\frac{1}{2}$, 1 and so on, which applies to ground pumice is also given to pumicite.

ORIGIN AND OCCURRENCE

Lava is a general term applied to molten rock forced out from volcanoes. Lavas range in composition from acid to basic, or from a rhyolite which consists mainly of quartz, orthoclase, microcline, or the acid plagioclase feldspars such as albite and oligoclase, to basaltic rocks which are mostly lacking in these minerals and are composed principally of labradorite and anorthite of the plagioclase feldspars, pyroxene, olivene, or other minerals less siliceous than those characterizing the so-called acid rocks.

Basic lava behaves differently from acid lava. Since a basic lava softens at a comparatively low heat and is characteristically liquid almost down to its freezing point, it will usually flow quietly from a volcano. Inasmuch as the internal prossure is relieved with little tendency for explosive eruption, pumice is rarely formed from basaltic lavas. Deposits of pumiceous basalts, however, do occur in the Sandwich Islands; such material is commonly yellow or brown in color, but may be pitch black when freshly exposed.

Acid lavas, on the other hand, melt at relatively high temperatures and are likely to be quite viscous when even slightly chilled. At temperatures which would have little effect upon a basic lava, they are apt to harden or cease to flow readily. Such stoppages in the earth's vents or safety valves (which volcanoes practically are) cause enormous pressures to build up until

the force becomes powerful enough to break through or blow out the plugging material. This may happen with such explosive violence as to hurl enormous masses of material out from the volcano, sometimes disrupting the mountain itself and spreading destruction for miles around. Examples of comparatively recent violent eruptions that have created vast quantities of pumiceous materials are those of Mt. Pelee on the island of Martinique, Krakatoa in the East Indies, Katmai on the Alaskan Peninsula, Vesuvius in Italy, and Mt. Colima on the west coast of Mexico.

During volcanic activity different conditions favor the formation of pumice, pumicite, and various extrusive or intrusive igneous rocks. Formation of pumice requires usually a molton acid lava, highly impregnated with gases, with conditions of temperature, pressure, and extrusion such as to permit the gas to expand in the hot lava forming bubbles or threads and fibers with thin glasslike walls between the cells. The vesicles must not expand to the point of disruption, and the walls must harden rapidly enough to prevent collapse upon any release of pressure and to prevent the formation of mineral crystals. Contamination with nonpumiceous materials such as rhyolite, felsite, scoriae, and obsidian are objectionable from a commercial viewpoint. Deposits of pumice are always found near centers of volcanic activity. Some occurrences are massive and capable of yielding pumice in lump form. Other deposits of pumice are quite loosely consolidated, being in the form of sand or gravel.

Slightly different conditions such as greater pressure and consequently a greater explosive force may produce pumicite or volcanic dust. In eruptions of this character a portion of the lava may be spumed into the air in very fine particles, not unlike pulverized glass, accompanied by water vapor, gases, and fragmental volcanic rocks. Much of the dustlike matter is borne hundreds and some of it thousands of miles by air currents. Heavy particles drop relatively near the source while the finer and more buoyant material is deposited at greater distances, so that a certain classification by particle size is accomplished.

The dust may settle in water or on land and may be reworked by wind and stream action. The loose, unconsolidated material behaves much like drifting snow or sand, and much of it is eventually carried away by streams. It may, however, drift into natural land depressions where it is subsequently buried. Deposits in the older geological formations are often too greatly contaminated with impurities to be of commercial importance at present. Somo deposits are found in old lake beds and may contain silt, clay, skeletons of diatoms, and other fossil material. Again they may be covered by lava flows or other volcanic formations, and where deposition has been in shallow seas, lakes, or ponds, the pumicite beds may become buried under sedimentary strata. Many of the commercial deposits even in Kansas, Nebraska, and Oklahoma are lens-shaped, grading laterally into sand or loess. They also contain islands of other material and are usually covered by loess, sand and gravel, or soil.

Extinct volcanoes in the Rocky Mountain region are thought to have been the source of pumicite in the States of the Great Plains region, and probably a number of volcanoes contributed to these deposits.

Pumiceous material is abundant in red clay found in ocean deeps. Although some may have been formed from submarine volcanoes, most of it probably was deposited on the surface of the ocean and sank as it became waterlogged. Floating deposits of such material, as much as 5 feet thick, have been reported after violent volcanic eruptions.

DEPOSITS IN THE UNITED STATES

Commercial deposits of pumice and pumicite occur in California, Kansas, Nebraska, Nevada, Oklahoma, and many other Western States. No attempt is made here to list all potential or undeveloped deposits of pumicite. Volcanic materials are abundant in many of the Western States, and vary in texture from the hard dense lava flows to the true pumice and pumicite. Much of the material is associated with tuffs of all kinds, which include the indurated pumicite, scoriae, and other materials peculiar to volcanic formations. The occurrences described in the following paragraphs include only those in districts from which production of pumicite has been reported to the bureau, or about which definite information is available as to characteristics.

Arizona. - Commercial deposits of pumicite occur in Yuma County near Bouse.

Arkansas-Louisiana. - Beds of volcanic tuff occur in the Cretaceous and Tertiary formations of this region. The material is encountered in deep wells and is useful only in so far as it forms key beds that aid in the proper correlation of other geological formations.

California. - Pumiceous material occurs in many counties in California, but pumicite is produced mainly from Fresno, Mono, Kern, and San Luis Obispo Counties. Siskiyou, Inyo, Mono, and Imperial Counties are the more important sources of pumice.

Colorado. - Deposits of pumicite of varying grades are abundant in the eastern part of Colorado.

Iowa. - A bed of pumicite 1 foot thick has been reported from Des Moines, occurring below the Wisconsin glacial drift and above the Peorian loess. This material appears to be only of scientific interest, perhaps indicating the eastern limits of the pumicite in the Great Plains region.

Kansas. - Extensive deposits of pumicite from 16 to 20 feet thick are worked in Meade County near Fowler. Over the pumicite is about 15 feet of sand and soil overburden. Other deposits occur in Sheridan, Norton, Rawlins, and Seward Counties. Minor occurrences have been found in many other counties of the State, some of which have yielded small tonnages.

Montana. - Pumicite occurs in many counties of Montana, the more extensive deposits being in Gallatin County where the material occurs in layers about 5 feet thick between impure beds of pumiceous material, the entire formation having a thickness of 100 to 1,000 feet.

Nebraska. - At one time Nebraska led in the production of pumicite, but the industry has lagged considerably since the development of the Kansas deposits. Pumicite is found in nearly every county of the State and, as in Kansas, coarse material occurs in the western part and fine material to the east. Most deposits are overlain by a mantle of loess which has been partly removed by erosion. Deposits are worked in Frontier County; other important deposits occur in Harland County.

Nevada. - Extensive deposits of pumicite occur in Nevada, but so far there has been little production from this State.

Oklahoma. - A number of deposits have been found in Oklahoma, but little if any pumicite is being produced at present. The largest occurrences are in the northeastern part of the State near Gate close to the boundary of Harper and Beaver Counties, where a thickness of 80 feet has been reported. Other deposits have been found west of Woodward, northeast from Custer City, and northwest from Wetunka, where a small tonnage has been produced. Buttram[7] reports deposits in Beaver, Harper, Woodward, Woods, Blaine, Custer, Kay, Okfuskee, Hughes, and Haskell Counties.

Oregon. - Pumicite occurs in eastern Oregon and important deposits are found in the valley of the Deschutes, Klamath, and Umatilla Rivers.

South Dakota. - A commercial deposit of pumicite occurs at Winner in Tripp County; it is utilized in the manufacture of powdered soap.

Texas. - A deposit of pumicite has been reported from Dickens County, 50 miles northwest from Double Mountain.

Washington. - Pumicite has been reported from the vicinity of various volcanic peaks such as Mount St. Helena.

Wyoming. - A deposit of pumicite 4 feet thick has been reported near Sportsman Lake 8 miles west of the station of Tie Siding on the Union Pacific Railway.

DEPOSITS IN CANADA

Extensive beds of pumicite occur in the vicinity of Waldeck, Saskatchewan, on the Canadian Pacific Railway, 11 miles northeast of Swift Current. British Columbia has reported pumicite from Dead Man's River, Arrow Lake, Haig, and Elko.

MINING

Pumice and pumicite are mined both by surface excavations and underground methods, but for operations handling a large tonnage, open-pit methods generally have been adopted. Where open-pit methods are employed the overburden, which usually is loess, sand, or soil, is removed by hand, team and scrapers, or power shovels. In some operations a thin layer of overburden is left to

7 - Buttram, Frank, Volcanic Dust in Oklahoma: Oklahoma Geol. Survey, Bull. 13, Dec., 1914, p. 34.

protect the underlying pumicite until it is mined. At other deposits the entire thickness is removed from as much of the area as may be required. The pumicite is then dug by pick and shovel, loaded into wagons, and carted to the mill or to the shipping point. Power shovels are used in some operations. Little or no blasting is required.

At the operations of the Pumicite Co., Fowler, Kans., overburden consisting of 8 to 10 feet of soil is removed with a 1-ton Lorraine gasoline shovel. Hand shovels are used to clean up the surface of the exposed pumicite, which is subsequently mined in much the same manner that the overburden is removed. Animal burrows, filled with dirt and clay, are found in the beds of pumicite and must be carefully separated.

The Fresno Estate Co., Fresno, Calif., with deposits on the McKenzie Ranch about 3 miles from Friant on the Southern Pacific Railroad, mines pumicite by drifts into the dome-shaped hills, which rise about 80 feet above the general ground level. These domes are virtually solid pumicite, except for about 4 feet of overburden. No explosives are used in the mining, which is done by pick and shovel.

PREPARATION FOR MARKET

Milling operations consist essentially in freeing the individual particle from one another, removing excess moisture, eliminating impurities such as clay, concretions and other harmful substances, grading the pumicite particles to the sizes desired, and packing for shipment.

At the pumicite deposits of the Middle West little or no crushing or grinding is necessary to free the pumicite grains. At the operation of the Cudahy Packing Co., in Meade County, the pumicite is shoveled by hand from the quarry floor into wagons, which transport the crude material to the mill. There it is dumped or shoveled into hoppers and then fed into an inclined oil-burning rotary kiln where the water, which may be as much as 20 per cent of the weight of crude material, is driven off. The dried pumicite is passed through a revolving 5-mesh screen to air separators. The oversize from the screen and the overweight from the air separators are rejected. Graded pumicite is packed in paper-lined freight cars and shipped to Chicago, where 2 per cent of soap is added to make a well-known cleansing compound.

At Fowler, Kans., the pumicite is taken from the pits to the mill (which has a capacity of 100 tons a day) and proceeds successively through a pan pulverizer, rotary oil-fired driers, Tyler Hum-mer screens, and a Bates packing machine. The product grades 90 per cent through 200 mesh, reject from the screens being returned to the pan pulverizer. Two rotary driers are used, each being 6 feet in outside diameter. One is 26 and the other 52 feet long. The inner compartment consists of a chamber the length of the drier, but 30 inches square in cross section. This shape has been found most suitable for handling and drying the pumicite, the square section having a tendency to break up the material and to prevent rotational slippage.

Bulk crude pumicite may be shipped in paper-lined box cars, but most of the prepared or refined material is packed in 3-ply paper sacks weighing 40 pounds each.

The La Rue-Axtell Pumice Co., Eustis, Nebr., mines its material in open pits. The first step in milling is to remove the moisture in rotary shell driers, both coal and distillate having been used for fuel. The material is subsequently screened to sizes varying from 40 to 90 per cent through 250 mesh. The pumicite is packed for shipment in paper bags of 75 pounds, burlap bags of 100, 150, or 200 pounds, or in barrels of 300 to 350 pounds capacity.

At Fresno, Calif., crude material is passed through a revolving cylinder equipped with paddles, which pulverize the lumps. This single mill operation produces a powdered product of which 98.8 per cent will pass a 325-mesh screen. Only one grade of pumicite is produced, most of which is used in concrete. Shipments are made in bulk and in 80-pound burlap sacks.

The California Quarries Corporation, with quarries at Tom, Mono County, Calif., mines pumice from a hill deposit with a gasoline shovel. The pumice occurs mixed with quartz sand which is removed by passing the crude material over dry concentrating tables. The separation is quite easily made because the pumice, due to its porous structure, is lighter than the sand. Pumice from the concentrating tables is screened to the desired sizes, the oversize being passed through a system of roll crushers. Pumice that is used with gypsum for plaster work is dried in rotary driers, packed in burlap sacks of 80 pounds each, and transported by an overhead tram to a warehouse on the railroad.

Virtually all sizes of pumice, from $\frac{1}{4}$ inch to minus 60 mesh, are produced. Most of the product is used in acoustic plaster and is sized to minus 8 and plus 30 mesh. Sizes plus 8 and minus 30 mesh are used for concrete work.

Most domestic pumice and pumicite is prepared for market in the Middle West or West, whereas much of the Italian pumice imported for domestic consumption is milled in the East. American-ground Italian pumice is prepared from small pieces of high-grade material (pezzame) which is shipped in bags to America for further preparation.

In the plant of the James H. Rhodes Co., at Dutch Kills Canal, Long Island City, N. Y., bags of pumice are transferred from lighters and dumped into a large elevator pocket. This material has already had a preliminary sorting and grading, impurities such as obsidian, particles of iron, and inferior grades of stone having been removed at the island of Lipari. After being elevated to storage bins the material is crushed, screened and subsequently carefully dried. The pumice is then sent through a series of bolting machines where it is sifted through Swiss silk and sized to the various desired grades. Patented machines are used to pack the material for shipment in barrels lined with crinkled paper.

DOMESTIC PRODUCTION

The following production figures of pumice and pumicite sold or used by producers in the United States, 1920 to 1930, have been compiled by Miss A. E. Davis of the United States Bureau of Mines.

Pumice and Pumicite Sold or Used by Producers in the United States, 1920-1930

Year	Short tons	Value	Value per short ton
1920	41,838	$114,433	$2.74
1921	37,108	158,540	4.27
1922	45,262	175,600	3.88
1923	56,575	214,169	3.79
1924	43,651	190,253	4,36
1925	40,380	179,020	4.43
1926	53,887	208,504	3.87
1927	53,298	221,481	4.16
1928	57,430	278,516	4.85
1929	67,013	353,064	5.26
1930	56,843	336,099	5.91

IMPORTS

More than 90 per cent of all pumice imported into the United States comes from Italy, where it is mined in the Lipari Islands just north of Sicily. Germany is second in importance, and imports from other countries are virtually negligible. Imports of pumice for consumption from 1919 to 1930 are given in the following tables:

Imports for consumption: Crude or Unmanufactured Pumice Valued at $15 a Ton or Less, 1919-1930

Year	Rate of duty	Quantity, long tons	Total value	Value per long ton
1919	5 per cent	4,044	$26,003	$6.43
1920	do	7,992	53,067	6.64
1921	do	2,216	15,180	6.85
1922 (Jan. 1-Sept. 21)	do	4,881	34,362	7.04
1922 (Sept. 22-Dec. 31)	1/10 cent per pound	2,516	17,714	7.04
1923	do	10,868	71,730	6.60
1924	do	7,995	70,977	8.88
1925	do	8,457	86,992	10.29
1926	do	8,371	85,141	10.17
1927	do	7,821	85,591	10.95
1928	do	9,206	94,804	10.30
1929	do	8,310	84,718	10.19
1930	do	5,570	55,014	9.88

Imports for Consumption; Crude or Unmanufactured Pumice Valued at More Than $15 a Ton, 1919-1930

Year	Rate of duty	Quantity, long tons	Total Value	Value per long ton
1919	5 per cent	1,190	$47,617	$40.01
1920	do	2,387	90,711	38.00
1921	do	371	13,569	36.57
1922 (Jan. 1-Sept. 21)	do	650	21,680	33.35
1922 (Sept. 22-Dec. 31)	¼ cent per pound	47	1,557	33.13
1923	do	125	5,970	47.76
1924	do	109	5,129	47.05
1925	do	598	11,600	19.40
1926	do	190	5,745	30.24
1927	do	151	4,783	31.68
1928	do	704	18,654	26.50
1929	do	323	11,056	34.23
1930	do	240	9,312	38.80

Imports for Consumption: Pumice Wholly or Partly Manufactured, 1919-1930.

Year	Rate of duty	Quantity, long tons	Total Value	Value per long ton
1919	¼ cent per pound	1,975	$44,692	$22.73
1920	do	3,948	103,683	26.26
1921	do	948	19,170	20.22
1922 (Jan. 1-Sept. 21)	do	1,954	31,095	15.91
1922 (Sept. 22-Dec. 31)	55/100 cent per pound	886	11,414	12.88
1923	do	2,504	48,194	19.25
1924	do	2,425	33,781	13.93
1925	do	2,670	30,873	11.57
1926	do	2,753	36,843	13.38
1927	do	2,009	32,019	15.94
1928	do	3,825	43,471	11.36
1929	do	4,051	46,838	11.56
1930	do (a)	3,158	29,673	9.40

(a) Effective June 18, 1930, the duty on wholly or partly manufactured pumice was advanced from 55/100 cent per pound to 3/4 cent per pound.

Imports for consumption: Manufactures of Pumice, or Manufactures of which Pumice is the Component Material of Chief Value, 1919-1930

Year	Rate of duty, per cent	Quantity, long tons	Total value	Value per long ton
1919	25	.....	$1,469	......
1920	25	.....	2,534	......
1921	25	.....	3,586	......
1922 (Jan. 1-Sept. 21)	25	.....	636	......
1922 (Sept. 22-Dec. 31)	35	9.38	836	$92.88
1923	35	18.32	2,709	152.78
1924	35	12.40	1,397	112.66
1925	35	4.77	995	208.59
1926	35	.....	1,720	......
1927	35	.....	1,681	......
1928	35	.....	2,507	......
1929	35	.....	1,332	......
1930	35	.....	388	......

TARIFF

The tariff act[8] of 1930 makes the following provisions for duties on pumice stone:

> Pumice stone, unmanufactured, valued at $15 or less per ton, one-tenth of 1 cent per pound; valued at more than $15 per ton, one-fourth of 1 cent per pound; wholly or partly manufactured, three-fourths of 1 cent per pound; manufactures of pumice stone, or of which pumice stone is the component material of chief value, not specially provided for, 35 per centum ad valorem.

ITALY

Vesuvius has been the source of much of the Italian pozzuolana cement, so named from the town of Pozzuoli near Naples. The cement was made in part from pumicite and was used extensively in constructing the famous Roman walls, aqueducts, and other historic structures. The Roman Pantheon, more than 2,000 years old, is a striking example of the durability of this kind of construction. The old Roman harbor works at Porto Trejano, Porto d'Anzio near Rome, and Porto Adriano near Bari are examples of this kind of cement used as hydraulic material. The practice of using pozzuolana materials was revived about 1882, and the results are said to have been uniformly satisfactory since 1888.

The pumice exported to the United States, however, comes from high-grade deposits on the barren and relatively inaccessible island of Lipari about 22

8 - Tariff Act of 1930, par. 206, p. 15.

miles from the northern coast of Sicily. The best material occurs along the eastern and northern coasts in the vicinity of Canneto where American-owned mills and warehouses are located.

High-grade pumice occurs in flat-lying veins or beds with related material known locally as "lapillo." Mining pumice by picks and shovels in excavations on the surface or underground by means of rude drifts and crosscuts, has been the occupation of the natives for many years and is their only means of livelihood at present. Little, if any, timbering is done, and individual workings are quite apt to cave before all available pumice can be removed. Men, women, and children engage in the work, toiling from 12 to 14 hours at wages that range from 20 to 40 cents a day. The pumice is loaded into baskets and carried to the women and children who sort the chunks into various grades.

Although some lump pumice is exported in the crude state, most of it is smoothed with files. Pieces smaller than 4 inches and sometimes larger ones of hard and inferior quality are rolled in heavy tin-plate cylinders. The highest grade lumps are wrapped in tissue paper, and packed with excelsior in wooden boxes and casks of 220 pounds (100 kilograms) or in barrels of 330 pounds (150 kilograms).

Smaller fragments of $\frac{1}{2}$ inch to $1\frac{1}{2}$ inches in size may be sold direct by miners to exporters. This material is commonly called "pezzame." It is packed for export in jute bags of from 88 to 176 pounds (40 to 80 kilograms). The greater part of it comes to the United States where it is ground for American trade. Some Italian ground pumice consists of very fine material screened in a primitive fashion at Lipari. A large proportion is of an inferior grade, consisting mainly of lapillo, although some is ground from smaller fragments of high-grade pumice.

At the mines, lapillo is dumped into long wooden chutes, from which it falls through screens. To be classed as powdered pumice, it must pass through a screen not larger than 20 mesh. Coarser than 5 mesh is grinding rock, while minus 5 and plus 20 mesh is termed "rasaglia," most of which is commonly rejected as being too small to handle profitably; some, however, is manufactured or ground. After the first rough screening the lapillo must be either sundried or heated over coal ovens to facilitate bolting. The bolting machines are hexagonal wooden wheels about 10 feet long and 3 feet in diameter and are equipped with wire screens of from 35 to 200 mesh. Hand power generally is employed for turning these wheels, but sometimes steam or oil engines are used.

Pumice is known by a number of local or Italian names applied to the various grades and sizes of lump material. Pomice grossa or grosse are the largest lumps from 4 to 8 inches in size. This grade is subdivided to fiore, quasi-fiore, and mordente, which are three grades of superior material; and bianche, dubbiose, and mere, which are three grades of inferior stone. Correnti is medium size and pezzame is smaller fragmental material. Bastardoni is medium lump and may be applied to very hard, heavy pumice usually 4 to 8 inches in size. Rasaglia and alessandrina are smaller pieces somewhat similar to pezzame. Pietra pomice in pizzi is the general term applied to lump pumice,

whereas pietra pomice polvere applies to powdered pumice stone. **Lapillo** is pumice dust or detritus, a fine powder occurring mixed with pozzame and rasaglia.

GERMANY

Trass is a cement made from a form of indurated pumicite occurring in the Eifel region of Germany. The material resembles soft stone which has been weathered considerably since deposition. In composition it is quite similar to pumicite of other localities.

Pumice sand and gravel occur in the Neuwied Basin in the Rhine Valley of Germany. The material is said to be very porous, light in weight, and to possess good insulating qualities. The composition varies from 55 to 70 per cent of SiO_2, 15 to 20 per cent of Al_2O_3, with about 14 per cent of potassium and sodium salts.

In the manufacture of building units the practice originally was to mix pumice sand with clay and milk of lime. The mass was then moulded and air-dried to make what were called floating stones; these were used locally at first, but in 1911 it is reported that 320,000,000 units were shipped to outside markets. The industry slumped during the World War but has been revived in recent years. Cement and quick-setting limes have largely replaced the cementing agents formerly used, and many other improvements have been made in manufacturing processes. To a certain extent the industry has felt competition from slag, clinker, and other materials.

According to an article appearing in Cement, Lime and Gravel, January, 1930, more than 600 factories produce various kinds of building material in which pumice is used. Automatic machinery is being installed which will make upwards of 3,000 bricks an hour. Many houses, offices, warehouses, factories, hospitals, barracks, churches, theatres, and other structures have been built with similar units. The following advantages are claimed: The material is of a highly porous character, very light, and resists frost. It is fireproof, and does not expand or contract appreciably, and because of its lightness and strength the steel framework can be lighter, which saves an expenditure of as much as 15 per cent in cost, to say nothing of the reduced cost for cartage and freight. On account of the lightness of the material a builder can handle and place in position at least 25 per cent more bricks a day.

The units are good sound insulators and are used largely for flooring with reinforced steelwork.

> Pumice stone is very porous and gives a natural ventilation to the walls of a building. No dry-rot or mouldy fungus is possible where pumice stone is used. Pumice-stone building material is square edged and gives a perfect bond for mortar or other binding agent. For plaster work no laths are required, and one coat of plaster is sufficient. Nails can be driven into pumice stone blocks and the material can be sawn. Pumice-stone building material will be found to have advantages for constructing agricultural

and fruit and farm buildings. The porous nature of the material gives natural ventilation, which allows fruit and farm produce being kept under improved conditions. Clean pure air is necessary for dairies and stables and it is claimed that cattle and horses are much more comfortable and healthy when housed in buildings made from pumice stone. There is no wall sweating with this new material.

AUSTRIA

Important deposits of pumice occur in the Tyrol district of Austria.

NEW ZEALAND

Deposits of pumicite as thick as 20 feet occur in southeastern New Zealand in the Gisborne district in the vicinity of Poverty Bay. Eardley-Wilmot[9] states that the annual production is between 2,000 and 4,000 tons.

JAPAN

Pumice stone used for boiler and furnace insulation, and known locally as koka seki, occurs on the Niijima Islands 90 miles south from Tokio.[10]

OTHER COUNTRIES

Volcanic material suitable for use in cement, occurring in southeastern France and the Azores, has been shipped to Portugal for river and harbor work and general construction. A pumicite, locally called tosca, is shipped from the Canary Islands to Spain. Santorin, a pumicite from the island of Santorin in the Grecian Archipelago, was used in the construction of the Suez Canal.

MARKETS AND PRICES

Domestic pumice and pumicite, and imported Italian pumice offer individual marketing problems because of differences in geographical source, physical properties, preparation for market, and utilization. Pumicite enters quite different markets from either lump or ground pumice, although there is an overlapping in certain forms of ground pumice stone used for scouring soaps, cleansing compounds, and a few other abrasive uses. As a rule, however, domestic pumicite is a different grade of material, commanding a lower price.

Certain of the largest pumicite consumers, manufacturing scouring and cleansing compounds, operate their own deposits and are seldom in the market for outside material. Pumicite used in large construction jobs is commonly sold on a contract basis. Only a relatively small quantity is sold through jobbers and dealers. Because of these conditions the open market for pumicite is relatively small and prices for sales are not available. From the value of production reported to the Bureau of Mines, however, a unit price per ton on all pumicite produced can be calculated. For 1930 this price was slightly less than $6.

9 - Eardley-Wilmot, V. L., Work cited, p. 91.
10 - Eardley-Wilmot, V. L., Work cited, p. 83.

As regards future trends, the only clues are furnished by the records of the past. During the last 10 years domestic production of pumicite has increased about 30 per cent in volume. Because of the constantly changing fabric of our civilization many old uses have diminished in importance, but such losses have been offset by the development of new uses. The use of pumicite in concrete and in the manufacture of cement may be of more importance in the future than it has in the past.

At present, marketing of pumicite is dependent quite largely upon specific demands, and new operators should study markets thoroughly before investing any considerable capital.

The growing demand for building materials possessing special properties for specific purposes perhaps offers one of the most promising potential markets for pumice and pumicite. American producers will do well to keep in touch with practice in Europe, where this branch of the industry has been developed much more intensively than in the United States.

Much of the domestic pumice, produced largely in California and Oregon, is ground and used similarly to pumicite. Some is used in various construction ways. Most of it, so far, has been utilized locally; it has not yet been able to penetrate successfully the ground-pumice markets in the industrial eastern United States, except for a short time during the World War when material from Italy was not available. During war times some consumers were able to use domestic supplies, whereas others found such material not up to the standards of the imported pumice. It is difficult to ascertain whether this dissatisfaction was due to differences in quality of raw stock, insufficient or perhaps careless preparation, or unjustified prejudice against an untried product. It is reported that a new attempt is being made to place western ground pumice on the eastern market in competition with imported material.

Pumice imports have remained fairly constant during the past 10 years. Most of the business has passed into the hands of a few manufacturers and dealers, resulting in higher standards of quality, better service to the consumers, and less confusion as to grade of material bought. Foreign pumice is of two kinds, American ground and Italian ground, the former being considered much superior in abrasive qualities, uniformity of grain size, and general utilitarian value.

American-ground Italian pumice which is pulverized, sized, and packed in bags in the United States sells for $30 or more per ton f.o.b. mill, New York, price depending upon grade and quality. Coarse sizes are higher in price than fine sizes. Prices for materials in bags are somewhat less than for that packed in barrels.

Italian-ground pumice is lower in cost than the American-ground Italian variety. One importer reports that Italian-ground material can be delivered c.i.f. Atlantic port for $10.50, which makes the delivered cost around $25.50 including the duty of $15 per short ton. On the other hand, pezzame or small

lump pumice such as is used as raw material for American ground pumice, costs about $18.50, including the duty of $2 per short tons. Other grades of lump pumice are still higher priced, running from 1.7 cents to 14.1 cents per pound, plus a duty of 25 cents for unmanufactured or 75 cents for manufactured pumice per 100 pounds. Prices vary according to quality and degree of preparation.

LIST OF PRODUCERS

The companies producing pumice and pumicite in the United States in 1930 and the locations of the deposits from which the pumice and pumicite were obtained were as follows:

Location of Deposit	Name and Address of Company
Arizona:	
Yuma County, Bouse	Yuma Products Manufacturing Co., 3648 Humboldt St., Denver, Colo.
California:	
Fresno County, near Friant	Earlonite Mining Co., Box 474, Selma, Calif.
Do.	The McKenzie Estate, Griffith-McKenzie Bldg., Fresno, Calif.
Imperial County, near Calipatria	A. W. Brand, 880 E. Colorado St., Pasadena, Calif.
Do.	Kalite Co. Ltd., South Oak Knoll Ave., Pasadena, Calif.
Imperial County, Niland	Flynt Silica & Spar Co., 1047 Richmond St., Los Angeles, Calif.
Inyo County, Shoshone	Chas. Brown, Shoshone, Calif.
Do.	California Talc Co., 837 Jackson St., Los Angeles, Calif.
Do.	Tonopah & Tidewater Railway, 1014 Central Bldg., Los Angeles, Calif.
Kern County, Saltdale	Cudahy Packing Co., 111 West Monroe St., Chicago, Ill.
Mono County, near Laws	California Quarries Corporation, 1300 Quinby Bldg Los Angeles, Calif.
San Bernardino County, Daggett	Hill Brothers Chemical Co., 2159 Bay St., Los Angeles, Calif.
San Luis Obispo County, near Paso Robles	M. L. Francis, R. F. D., Creston, Calif.
Kansas:	
Grant County, Santanto	H. H. Zimmerman, Adams, Kans.
Meade County, Fowler	Cudahy Packing Co., 111 West Monroe St., Chicago, Ill.
Do.	The Pumicite Co., 4025 Clara Ave., St. Louis, Mo.
Norton County, Calvert	The Davidson Pumice Co., Norton, Kans.

Nebraska:

Frontier County, Eustis	La Rue-Axtell Pumice Co., Eustis, Nebr.

South Dakota:

Tripp County, Winner	Klenit Corporation, Winner, S. Dak.

LIST OF BUYERS

The following are some of the more important buyers of pumice and pumicite particularly for abrasive use:

Los Angeles Soap Co., 617 East First St., Los Angeles, Calif.
Allied Industrial Products Co., 124 N. May St., Chicago, Ill.
Cudahy Packing Co., 111 West Monroe St., Chicago, Ill.
Holman Soap Co., 3104 Fox St., Chicago, Ill.
Matchless Metal Polish Co., 842 W. 49th Place, Chicago, Ill.
Mineral Soap Manufacturing Co., Lowell, Mass.
Leon Hirsh & Son, 368 Greenwich St., New York, N. Y.
Charles B. Chrystal Co., (Inc.), 11 Cliff St., New York, N. Y.
Stanley Dogget (Inc.), 1 Cliff St., New York, N. Y.
A. Klipstein & Co., 11 Darrow St., New York, N. Y.
Whittaker Clark & Daniels (Inc.), 245 Front St., New York, N. Y.
Mr. Jerome Alexander, 50 East 41st St., New York, N. Y.
Rome Soap Manufacturing Co., Rome, N. Y.
Larkin Co. (Inc.), 680 Seneca St.,Buffalo, N. Y.
The DuBois Soap Co., 1120 W. Front St., Cincinnati, Ohio.
National Sales (Corporation), 31-35 East 13th St., Cincinnati, Ohio.
Chas. W. Young & Co., 1247 No. 26th St., Philadelphia, Pa.
Charles A. Wagner Co. (Inc.), 814 Noble St., Philadelphia, Pa.

LIST OF IMPORTERS

The following firms are among the more important importers of pumice, and due acknowledgment is made for their cooperation in assembling information on foreign material.

James H. Rhodes & Co., 153 W. Austin Ave., Chicago, Ill.
Charles B. Chrystal (Inc.), 11 Cliff St., New York, N. Y.
K. R. Griffiths & Co. (Inc.), 110 East 42nd St., New York, N. Y.
Hammill & Gillespie (Inc.), 225 Broadway, New York, N. Y.
Whittaker Clark & Daniels (Inc.), 245 Front St., New York, N. Y.

I. C. 6561

March, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

QUARTZ GEM STONES

BY

I. Aitkens

I.C. 6561
March, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

QUARTZ GEM STONES 1/
(Amethyst, Carnelian, Rock Crystal, Rose Quartz, Smoky Quartz, etc.)

By I. Aitkens 2/

CONTENTS

INTRODUCTION

Quartz maintains a rather conspicuous position among gem stones, despite its being the commonest and, in its natural form, the most easily recognized of mineral substances. This prominence is due primarily to the extensive use of amethyst, which provides stones of the finest violet color, and also to the wide employment of yellow quartz (citrine) which is called topaz by jewellers and vies with the true topaz in color and brilliance. Unlike many less costly gem materials quartz is hard and durable.

DESCRIPTION AND PROPERTIES

Quartz crystallizes in the rhombohedral system, generally as 6-sided prisms terminated at each end by 6-sided pyramids. Owing to differences in color and texture, some of which are of minor importance, quartz has been given a great variety of names. A considerable number of these names

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6561."

2/ Rare metals and nonmetals division, U. S. Bureau of Mines.

accompanied in each case by a brief description will be found in the appendix. The gem varieties of quartz, however, are all included in the following groups:

Phanerocrystalline	Cryptocrystalline
Amethyst	Agate
Aventurine (quartz)	Agatized wood
Cat's eye	Bloodstone
Citrine	Carnelian
Gold Quartz	Chalcedony
Milky quartz	Chrysoprase
Rock crystal	Egyptian jasper
Rose quartz	Eye agate
Rutilated quartz	Fortification agate
Siderite	Jasper
Smoky quartz	Moss agate
Tiger's eye	Onyx
	Plasma
	Prase
	Riband jasper
	Sardonyx

The composition of quartz is essentially silica or silicon dioxide, SiO_2. When pure it is colorless, but the mineral is to be found in practically every color, due to impurities. These colors, however, are unstable and may be destroyed by a low heat. They are also susceptible to change by the action of X-rays, ultra-violet light, and radium. The luster is vitreous and the mineral may be transparent, translucent, or opaque.

Quartz is uniaxial and optically positive. Its mean index of refraction is only 1.55; its double refraction 0.009; and its dispersion 0.013; all are rather low, as gem materials go. The fracture is pronounced conchoidal. Quartz has a hardness of 7 and is very durable. While the specific gravity of pure quartz is 2.65 to 2.66, the range for its many varieties is from 2.5 to 2.8.

USES

Many varieties of quartz are used in the cheaper grades of jewelry, but with the notable exception of the better grades of amethyst, which are very beautiful, most quartz material is worth little more than the cost of cutting. In the form of clear rock crystal it is used, however, for innumerable ornamental objects such as cups, vases, art figures, crystal balls, etc., and is often cut into beads and fancy shapes. Fashioned to resemble the diamond, it is sold as "rhinestone," "Herkimer diamond," "Lake George diamond," or "Cornish diamond." Rock crystal also furnishes the material for certain special glasses and fused silica ware; and selected crystals are employed in optical instruments, for scientific and military use, chiefly as wedges for microscopic

work and as spectographic prisms, and for piezo-electric plates which serve to control wave lengths in radio broadcasting.

The best grade of rose quartz is used in the manufacture of jewelry, chiefly for pins, necklaces, and settings for rings; however, it is also used extensively for vases, trays, paper weights, and like ornaments, and the cheaper grades are used for monumental works, such as facings for fireplaces and water fountains, and in pebble-dash finish in stucco work.

IDENTIFICATION

Quartz may be distinguished by its glassy luster, conchoidal fracture, hardness, and crystal form. Its hardness is such that it will scratch glass. Its lack of cleavage, general glass-like appearance, and pronounced conchoidal fracture, afford an easy determination. It is readily distinguished from topaz because it is softer and lighter, it will float in methylene iodide while topaz sinks rapidly.

SUBSTITUTES

Small quartz crystals have been produced artificially but never in sizes or colors to rival the natural mineral. Moreover, since most varieties are worth little more than the cost of cutting, there is no commercial advantage in manufacturing artificial material. Instead of having substitutes, quartz itself is used widely in imitation of the more expensive gems. Citrine, or yellow quartz, and smoky quartz, for example, are sold under trade names such as Scotch topaz, false topaz, and smoky topaz.

HISTORY

Quartz in its various forms has attracted attention from the earliest times. The water-clear crystallized variety was known to the Greeks as krystallos meaning clear ice; hence the name "crystal" or "rock crystal" as now applied to this particular variety. The name quartz[3] is an old German mining term of unknown meaning which has been in general use in all languages since the sixteenth century.

Amethyst is a name derived from the Greek word meaning not drunken--possibly due to a notion that it would protect the wearer from the results of unrestrained libations.

Chalcedony derives its name from the Greek name of a town in Asia Minor.

Prase, due to its dull leek-green hue, takes its name from the Greek word meaning a leek.

Plasma, which may have the same derivation, is a brighter leek-green.

3/ Smith, G. F. Herbert. Gem Stones: London, 1912, p. 240.

MODE OF OCCURRENCE

Quartz is widely distributed and occurs in rocks of all ages and of nearly every type, whether igneous, sedimentary, or metamorphic. Next to the feldspars, it is the most abundant mineral in the crust of the earth, forming about 12 per cent of the entire lithosphere.4/

MINING

Quartz is generally mined from open pits. After being blasted from the ledge the material is broken up with sledges into lumps 6 or 8 inches in size and the quartz material is then sorted from the other mineral substances.

In Montana and those States through which the Yellowstone and Missouri Rivers flow, agates are found along the river beds and sometimes on gravel bars quite a distance from the sources of these rivers. Ranchers, sheep herders, and others living in these districts gather them, some making agate gathering a regular business. The surface stones have been practically all picked up and present supplies are limited mainly to stones exposed by the erosion produced by winter ice and spring rains.

DOMESTIC PRODUCTION

The production in the United States of most varieties of quartz of gem quality has always been rather small. Only a few domestic varieties are worth more than the cost of cutting, and then only when found as exceptionally fine stones. The following table gives the value of gem quartz produced in the United States from 1906 to 1921, when the United States Geological Survey discontinued the canvass of producers:

Table 1. - Value of quartz gems produced in the United States, from 1906-1921 1/

Year	Value	Year	Value	Year	Value	Year	Value
1906	$7,050	1910	$3,872	1914	$18,838	1918	$15,211
1907	8,955	1911	3,834	1915	35,724	1919	17,632
1908	4,163	1912	3,313	1916	25,707	1920	14,676
1909	5,659	1913	16,861	1917	28,273	1921	11,114

1/ Compiled from Mineral Resources of the United States, Part 2 (annual chapters).

Although production in the United States is of small consequence, in certain cities the shops display cut gems and polished slabs of unusually high value. Billings, Livingstone, Miles City, and Glendive are the cutting centers in the State of Montana. Unusual specimens are prepared which bring as much as $250. Large quantities of this material are polished and mounted for eastern trade at Portland, Ore., and Seattle, Wash. The finish given these domestic stones has proved that the craftmanship of American workmen is equal to that of the foreign expert.

4/ Clark, F. W., Bull. U. S. Geol. Surv., 228, 1904, pp. 19, 20.

DEPOSITS IN THE UNITED STATES

In the past many fine agates were found on the lake shores of the Upper Peninsula of Michigan, where they weathered out of the trap rock. Amethyst also has been produced in various localities of the United States. In general, however, the domestic output has been small and sporadic, individual deposits usually being soon exhausted or proving unprofitable to work on a commercial scale.

Agates 5/

In many localities in the United States agates of considerable beauty if not of great size are found. Those found in Michigan on the shores of Agate Bay, Lake Superior, have rich coloring and make attractive charms and ornaments. Agates are found in the stream beds of many rivers in Colorado, Montana, and other regions of the Rocky Mountains. They are found also along the Mississippi River, especially in Minnesota, and along the Fox River in Illinois, in the trap rocks bordering the Connecticut River, and on the coasts of California and Oregon. The variety found in California is cut and sold under the name "California moonstone," a term as misleading as most trade names.

Moss or Landscape Agate

According to Foshag,6/ when nature desires to produce a unique type of agate, she permeates the silica with manganese or iron compounds, giving rise to moss or treelike forms, or islands in a minature ocean. This is designated as moss or landscape agate. Montana and Wyoming are the chief American sources; Montana supplies much of the moss agate cut for jewelry from pebbles and cobbles found along the Yellowstone River and its tributaries and on the mesas and buttes for many miles away from the river. No one knows just when or how these agates originally developed, but here they occur in rounded cobbles covered sometimes with a chalk-like but hard coating of silica.

In Laramie County, Wyoming, moss agate occurs in an irregularly shaped vein varying from less than an inch to nearly 2 feet in thickness and cutting nearly vertically across limestones of carboniferous age. The Wyoming agate is a clouded and more opaque form than the Montana agate.

Amethyst

Amethystine quartz has a wide distribution in the eastern United States, but nowhere does it occur in sufficient quantity to afford a stable industry. The crystals are sometimes of a beautiful color but are almost invariably scarred and broken.

5/ Farrington, O. C., Agate--Physical Properties and Origin: Field Mus. of Nat. Hist., Geol. Leaflet 8, Chicago, 1927, p. 5.
6/ Foshag, W. F., Gems and Gem Minerals: Smithsonian Sci. Ser., vol. 3, pt. 2, 1929, p. 229.

About 1921, amethysts were found and mined in Arizona to a notable extent on claims near the Roosevelt Dam. This was the first amethyst deposit of commercial importance discovered in the State. Although these crystals were taken from only 20 inches below the surface, they were of such fine quality that orders to the amount of $2,000 were obtained almost immediately from dealers and lapidaries in Los Angeles.

Amethyst Harbor on the shores of Lake Superior has yielded very large crystals, though not of good gem quality. Other localities that have produced good stones are Providence Township, Delaware County, and Chester County, Pa.; Haywood County, N. C.; and Rabun County, Ga.

Chrysoprase

According to Kunz,7/ two claims of chrysoprase were located about 6 miles southeast of Ione, Calif. At this location the chrysoprase occurs in serpentine as a seam 1-1/5 inch wide.

Crystal Quartz

In the eastern United States and in the Rocky Mountain regions, well-crystallized colorless quartz occurs in association with pegmatite veins. Beautifully clear examples of this crystal quartz have come from the Hot Springs region of Arkansas, where the mineral occurs as a secondary product in clefts and crevices in sandstones. This region forms one of the world's most noted deposits of crystal groups, where quartz occurs in small, very perfect, and highly lustrous crystals.

In the cavities of the limestones of Herkimer County, N. Y., are occurrences of small, very perfect, and highly lustrous crystals. A peculiarly interesting characteristic of this particular deposit is that the crystals are rarely over a centimeter in diameter; as a consequence, the natural crystals are sometimes perforated and strung as beads without further preparation, and prove very effective as well as unique; so great is their brilliance that they are commonly called "Herkimer diamonds."

Onyx 8/

An important deposit of onyx was discovered in Madison County, Mont., during the autumn of 1929, which is said to rival the Egyptian stone in both color and design. The Montana Onyx Co. of Butte first opened this quarry and is taking out the stone in small quantities for cutting and polishing. This onyx lies upon the surface in well-marked ledges, in a vein that has been traced over a quarter of a mile, and comes in a multitude of colors which form the most beautiful tapestry designs. Approximately one hundred different colors have been observed by the owners of this quarry.

7/ Kunz, Geo. F., Gems and Precious Stones: Min. Ind., 1927, p. 509.

8/ Pit and Quarry. Huge Deposit of Onyx found in Montana. Vol. 20, No. 12, Sept. 10, 1930, pp. 19-20.

Rose Quartz

What is declared to be one of the largest deposits of high-grade rose quartz in the country, if not the world, is located about 8 miles from Custer, S. Dak.9/ Throughout this district the quartz is of a beautiful rose color and is found in large segregations and also in small scattered masses. This district produces all gradations in color from white to pale pink or deep rose. At the Scott rose quartz mine near Custer the quartz occurs in large irregular masses in pegmatite, and it is believed that it represents a late crystallization during the consolidation of the magmatic solution at temperatures somewhat below the temperatures at which white quartz crystallized.

FOREIGN LOCALITIES

Brazil is noted for very important deposits of clear crystal quartz as well as of quartz of other varieties. Much of the quartz used for spectacle lenses and other technical purposes comes from Brazil, especially from the States of Minas Geraes, Sao Paulo, and Goyas.

In the Swiss Alps both clear and smoky crystals occur in open cavities. From one such druse 20,000 pounds of good crystals were obtained. Lately Madagascar has produced much good rock crystal, which is utilized mostly in the manufacture of beads.

Agates 10/

At present the most extensive occurrence of agate known is in the mountain chain extending from Porto Alegre, State of Rio Grande do Sul, in Brazil, to the district of Salto in northern Uruguay. These agates form the principal source of supply for commercial purposes and are said to surpass all others in size and beauty.

Associated with the agates in the agate-bearing area in Uruguay are many hollow stones which are often lined with amethyst crystals. The agates are dug from shallow pits in grazing lands, where they occur in considerable quantity. They are also found in the stream beds of the region, where they have been deposited by floods. At present the largest diggings are about 100 miles north of Salto.

Amethyst

A discovery of amethyst was made in the vicinity of Pretoria, in South Africa, about 1927, where material of unusually deep color was found in large quantities. Although some 6,000 pounds of bulk material has been shipped, only specimens of jewel grade have been sent out. In 1928, a syndicate was formed with the object of exploiting this new discovery. During that year a 215-pound amethyst was found near Bahia, Brazil, and was sent to Oberstein, Germany,

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9/ Rocks and Mineral, Rose Quartz: Vol. 4, June, 1929, p. 43.
10/ Farrington, O. C., Work cited, pp. 4-6.

to be worked up. This was not of jewel grade, however, and would have been more valuable as a specimen.

Rock Crystal

In certain localities of Brazil, rock crystal is found in deposits that are usually small and scattered, but the crystals are water clear and range in size from 4 to 10 pounds. During the World War tons were produced to supply the demand for lenses, etc. More recently the demand for optical material has been greater than ever, and it is a question whether this material has ever before been so extensively used in physical experiments.

Recently two great masses of rock crystal, one weighing 528 pounds and the other 375 pounds, have been discovered in Brazil. Although the demand and price have dropped considerably in late years, rock crystal for spectacles and other optical work has increased on the average and the State of Bahia now supplies an abundance of it for these purposes.

In 1926, 14,016 pounds of rock crystal valued at $6,704 was exported from Bahia; in 1927, exports increased to 28,958 pounds valued at $10,072, a gain of 100 per cent in volume and of 50 per cent in value. However, exports dropped in 1928 to 19,408 pounds valued at $8,581.

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APPENDIX: NAMES OF SPECIES AND GEM VARIETIES

Names of Mineral Species and Gems of the Quartz Group

Name	Description
Agate	Variegated chalcedony.
Agate jasper	Intermediate between jasper and chalcedony with predominant translucent chalcedony; jasper with bands of chalcedony.
Alaska diamond	Quartz.
Alençon diamond	Quartz crystal from Alençon, France.
Aleppo stone	Eye agate.
Amberine	Yellowish-green agate from the Death Valley region, Calif.
Amethyst	Purple and bluish violet quartz, in crystals.
Amethystine quartz	Quartz of an amethyst color, not necessarily in crystals.
Anacona ruby	Quartz.
Apricotine	Yellowish red quartz pebbles from vicinity of Cape May, N. J.
Arkansas diamond	Quartz crystals from Arkansas; also diamond from Arkansas.
Arrow points	Indian arrowheads mostly made of quartz, more rarely of obsidian or other fine-grained rock.
Aventurine	Opaque yellow, brown, or red massive quartz containing inclusions of minute scales of some other mineral, such as mica or iron oxide.
Azure quartz	Blue quartz.
Baffa diamond	Quartz crystal.
Basanite	Velvet black, flinty quartz.
Beckite	Silicified coral shells or fossiliferous limestone replaced by silica.
Beekite	Beckite.
Bishop's stone	Amethyst.
Bloodstone	Massive dark green jasper (plasma) with red or blood-colored spots; also hematite (German usage).
Blood jasper	Bloodstone.
Blue chrysoprase	Chalcedony stained blue with chrysocolla.
Blue moonstone	Blue chalcedony from the Death Valley region, Calif.
Bohemian diamond	Rock crystal (quartz).
Bohemian ruby	Red or rose quartz.
Bohemian topaz	Yellow quartz.
Brazilian diamond	Clear quartz from Brazil; also diamond from Brazil.
Brazilian pebble	Rock crystal (quartz).
Brazilian topaz	Smoky quartz artificially changed to yellow by heat; also golden to reddish yellow topaz.

Names of mineral species and gems of the quartz group (Cont.)

Name	Description
Briançon diamond	Quartz crystal from southeastern France, cut in Briançon.
Bristol diamond	Quartz crystal from Cornwall, England.
Burnt amethyst	Purple amethyst changed to brownish yellow by heat.
Cairngorm	Yellow to smoky brown, gray, or black quartz.
California moonstone	White or gray chalcedony.
Cape May diamond	Colorless and clear quartz crystal from Cape May, N. J.
Carnelian	Translucent red chalcedony.
Carnelian-onyx	Agate with red and white bands.
Catalinite	Beach pebbles from Santa Catalina Island, Calif.
Catalina-sardonyx	Catalinite.
Cat's-eye	Chatoyant quartz; or any mineral having a changeable luster or showing opalescence without play of colors.
Cer-agate	Chrome-yellow agate from Brazil.
Chalchihuitl	Applied to prase, green jasper, emerald, turquoise, and jadeite from Mexico.
Chalcedony	Compact silica, transparent or translucent, with a waxy luster.
Chalcedony-onyx	Agate with white and pale bands.
Chalcedonyx	Chalcedony with alternating stripes of gray and white.
Chert	Compact silica, includes flint, hornstone, and jasper.
Chinarump	Petrified wood from Arizona.
Citrine	Golden yellow quartz.
Cloudy chalcedony	Chalcedony with dark cloudy spots in a light gray transparent base.
Colorado topaz	Citrine (yellow quartz); also topaz from Colorado.
Cornish diamond	Quartz crystal from Cornwall, England.
Cotterite	Quartz having a metallic pearly luster.
Creolite	Banded jasper from Shasta County, Calif.
Crispite	Sagenite.
Crystal	Colorless transparent quartz; also artificial flint glass.
Cupid's darts	Quartz crystal with needle-like inclusions of goethite.
Dauphine diamond	Rock crystal (quartz).
Dendritic agate	Mocha stone and moss agate.
Egyptian jasper	Banded yellow, red, brown, or black jasper.
Egyptian pebble	Egyptian jasper.
Eldoradoite	Iridescent quartz from Eldorado County, Calif.

Names of Mineral Species and Gems of the Quartz Group (Cont.)

Name	Description
Emeraldine	Chalcedony artificially colored green.
Enhydros	Hollow nodules of chalcedony partly filled with water.
Eye agate	Concentric rings of agate with a dark center, also thomsonite.
False diamond	Quartz crystal.
False lapis	Agate or jasper artificially colored blue.
False topaz	Yellow quartz; also yellow fluorite.
Fancy agates	Agates showing delicate markings and intricate patterns.
Fleches d'amour	Sagenite (quartz).
Fleurus diamond	Quartz crystal.
Flint	Compact silica, opaque and of dull colors.
Flower stone	Beach pebbles (chalcedony) with flower patterns.
Fortification agate	Agate with parallel zigzag lines.
Fossil coral	Coral replaced by silica (beckite).
Frost stone	Translucent gray chalcedony with pure white patches or tufts like snowflakes, scattered through it, from the Mohave Desert, Calif.
Gold quartz	Massive quartz inclosing gold.
Golden topaz	Golden yellow citrine (quartz); or topaz of a golden yellow color.
Hair stone	Quartz with inclusions of hair-like crystals or fibers of some other mineral. Same as sagenite.
Heliotrope	Bloodstone (quartz).
Herkimer diamond	Clear quartz crystal from Herkimer County, New York.
Horatio diamond	Colorless quartz from Arkansas.
Hornstone	Compact form of silica, like flint but more brittle.
Hungarian cat's-eye	Quartz cat's-eye.
Hyacinth	Red zircon; also wrongly applied to essonite or other light-colored garnets, to yellowish red spinel from Brazil, and to red iron-stained quartz.
Hyacinth of Compostella	Quartz, with red hematite inclusions.
Imperial yu-stone	Green aventurine quartz.
Indian agate	Moss agate.
Indian topaz	Saffron-yellow topaz; also yellow quartz.
Iolanthite	Jasper from Crooked River, Crook County, Oregon.
Iridescent quartz	Rock crystal (quartz) filled with fine cracks containing air films which reflect the colors of the rainbow.

Names of Mineral Species and Gems of the Quartz Group (Cont.)

Name	Description
Iris	Iridescent quartz; also applied to other iridescent minerals. California iris is spodumene.
Irish diamond	Quartz crystal from Ireland.
Isle of Wight diamond	Quartz crystal.
Jasp agate	Intermediate between jasper and chalcedony with predominant opaque jasper.
Jasper	Massive quartz, impure and opaque, containing more iron oxide than agate.
Jasperine	Banded and variously colored jasper.
Kinradite	Jasper with spherulites of quartz, from the region around San Francisco, Calif.
Lake George diamond	Clear quartz crystal from Herkimer, N. Y.
Lavendine	Amethyst (quartz).
Love arrows	Sagenite (quartz).
Lydian stone	Basanite (quartz).
Maderia topaz	Citrine (quartz).
Marmorosch diamond	Quartz crystal from Marmaros Comitat, Hungary.
Milky quartz	Milky white in color and nearly opaque. Sometimes with greasy luster. (Dana).
Mocha stone	Chalcedony with brown, red, or black, tree-like inclusions of manganese oxide.
Mohave moonstone	Translucent, lilac-tinted chalcedony from the Mohave Desert, Calif.
Montana agate	Moss agate from Montana.
Mont Blanc ruby	Quartz.
Moonstone	Commonly but erroneously applied to some white or gray chalcedony and to satin spar (gypsum).
Mora diamond	Probably quartz crystal.
Morion	Deep black almost opaque smoky quartz.
Moss agate	Chalcedony with greenish moss-like or tree-like inclusions.
Mother-of-emerald	Prase (quartz).
Myrickite	Agate or chalcedony containing bright red inclusions of cinnabar, from the Death Valley region, Calif.
Needle stone	Sagenite (quartz).
Nicola	Onyx with a black or brown base and a bluish white, thicker, wavy top layer.
Novaculite	Fine-grained hard sandstone; flint (quartz).
Occidental agate	Agate less perfect than oriental agate.
Occidental amethyst	True amethyst (quartz).
Occidental cat's-eye	Quartz cat's-eye.
Occidental chalcedony	Somewhat opaque chalcedony; more opaque than oriental chalcedony.
Occidental diamond	Rock crystal (quartz).
Occidental topaz	Yellow quartz.
Onegite	Quartz with inclusions of hair-like crystals of geothite.

Names of Mineral Species and Gems of the Quartz Group (Cont.)

Name	Description
Onyx	Banded chalcedony with alternating bands of cloudy milk-white and another color, usually black.
Orange topaz	Smoky quartz changed to yellow by heat; or Spanish topaz.
Oriental agate	Finely marked and very translucent agate.
Oriental chalcedony	Very translucent chalcedony. (compare with occidental chalcedony).
Oriental jasper	Bloodstone (quartz).
Ouachita stone	Novaculite (whetstone); quartz.
Paphos diamond	Quartz.
Pebble	Rock crystal (quartz).
Pecos diamond	Quartz from Pecos River, Texas.
Petrified wood	Wood replaced by silica.
Plasma	Massive translucent quartz, dark grass-green in color, sometimes with white or yellow inclusions of celadonite or of delessite.
Prase	Massive, translucent, and spotted quartz of a green to leek-green color caused by inclusions of minute crystals of actinolite or other minerals.
Prismatic moonstone	Cloudy chalcedony (quartz) from Mohave Desert, Calif.
Pseudodiamond	Quartz crystal.
Quebec diamond	Quartz crystal.
Rainbow agate	Agate which shows iridescence when cut across the concentric structure.
Rainbow quartz	Iridescent quartz.
Rhinestone	Rock crystal (quartz).
Riband agate	Agate with parallel layers.
Riband jasper	Jasper with differently colored, alternating bands.
Ribbon agate	Banded agate.
Ring agate	Agate with differently colored bands arranged in concentric circles.
River agate	Moss agate pebbles found in brooks and streams.
Rock crystal	Clear quartz crystal.
Rose quartz	Massive rose-red to pink quartz.
Rubasse	Quartz artificially stained red.
Sagenite	Transparent quartz with inclusions of hair-like or needle-like crystals or fibers of some other mineral, generally rutile.
Sandy sard	Sard dotted with darker spots (quartz).
Sapphire quartz	Blue quartz.
Sapphirine	Blue chalcedony, blue quartz; also blue spinel.
Sard	Chalcedony of a rich brown color, with a reddish tint; brownish red or dark brown carnelian (sardoine).
Sardoine	Brownish red or dark brown carnelian.
Sardonyx	(Sard-onyx) white and brown banded chalcedony.

Names of Mineral Species and Gems of the Quartz Group (Cont.)

Name	Description
Saxon topaz	Pale wine-yellow topaz; also citrine (quartz).
Schaumberg diamond	Quartz crystal from Schaumberg, Hesse, Germany.
Schiller quartz	Quartz cat's-eye.
Scotch topaz	Smoky quartz.
Semicarnelian	Yellow agate.
Siberian amethyst	Rich or dark colored amethyst.
Siderite	Sapphirine (blue quartz).
Silicified wood	Wood replaced by silica and small amounts of iron compounds.
Sinople	Quartz having red hematite inclusions.
Smoky quartz	Quartz crystals of a smoky or brown color.
Smoky topaz	True topaz of a smoky color; also more commonly smoky quartz.
St. Stephen stone	Translucent chalcedony with round blood-red spots through it.
Star stone	Starolite (quartz).
Starolite	Asteriated quartz.
Swiss lapis	Agate or jasper artificially colored blue.
Test stone	Basanite (jasper).
Texas agate	Agate jasper from Texas.
Thetis hairstone	Transparent quartz with inclusions of hair-like crystals of green actinolite.
Tiger-eye	Yellow to brown altered crocidolite.
Touchstone	Basanite (jasper).
Tree agate	Mocha stone.
Tree stone	Mocha agate.
Trenton diamond	Quartz crystal from Herkimer County, N. Y.
Unripe diamond	Quartz.
Vallum diamond	Quartz crystals from the Tanjore district, Madras Presidency, India.
Venus's hairstone	Sagenitic quartz (Fay).
Violite	Compact purple chalcedony from San Diego County, Calif.
Water agate	Shell of chalcedony containing bubbles of water.
Wax agate	Yellow agate, with a pronounced waxy luster.
White carnelian	Cloudy, milk-white, or very pale reddish or yellowish chalcedony.
White sapphire	Colorless quartz or corundum.
White topaz	Colorless quartz or topaz.
Wood agate	Wood petrified or replaced by agate.
Wood stone	Silicified wood.
Zonite	Variously colored chert or jasper, from Arizona.

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UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

SOAPSTONE

BY

H. Herbert Hughes

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SOAPSTONE[1]

By H. Herbert Hughes[2]

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INTRODUCTION

Soapstone commonly is associated with talc, and reports on either commodity usually contain a discussion of the other. From a geological or mineralogical standpoint the combination is justifiable, for soapstone essentially is impure talc. Commercially, however, the relation between them is remote. About 95 per cent of all talc sold in the United States is in ground form, whereas practically all soapstone is sold as furnace blocks or as fabricated structural slabs for many and diverse applications.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6563."

2 - Assistant mineral economist, building materials section, U. S. Bureau of Mines.

Most of the ground talc consumed in the United States is sold for purposes where the white color, slip, fineness of grinding, and lack of grit give the product decided advantages over other pulverized rocks. Some waste soapstone from fabricating plants is ground and marketed in competition with talc. This condition, however, exists only when the material is to be used for nonspecialized purposes. Furthermore, in these markets soapstone is in direct competition with ground limestone, whiting, slate flour, and similar products.

Many deposits of high-grade talc contain local segregations of impurities which lower the quality of the material and make it unsuitable for sale as first-quality talc. This product is comparable to ground soapstone, and there is some overlap in this market between talc and soapstone. This overlapping, however, is insignificant when contrasted with the total volume of either talc or soapstone production.

The soapstone industry in the United States is largely in the hands of one company, the Virginia Alberene Corporation, with mills and quarries at Schuyler, Va. The purpose of this paper is to describe briefly the properties, uses, and methods of quarrying and manufacture of soapstone, as well as general conditions in the industry. The help of the officials of the Virginia Alberene Corporation is acknowledged.

DEFINITION

Theoretically, talc is a hydrous metasilicate of magnesium with a definite chemical composition, expressed by the formula $H_2Mg_3(SiO_3)_4$. Actually, however, the silica or magnesia content may vary as much as 3 or 4 per cent with little change in the appearance of the mineral. Iron oxide is the principal impurity. Talc commonly is called steatite or soapstone, but these terms apply more accurately to modifications of true talc. Steatite usually is defined as massive talc that may be used in block form; French chalk is the most common example. Soapstone, in its original sense, apparently was synonomous with steatite. Agalmatolite is a name given to both massive talc and massive pyrophyllite, but it more properly should be reserved for the latter.

At present, proper usage demands that the term "soapstone" shall include all massive gray to bluish or greenish talcose rocks which, with few exceptions, have a soapy feel and can be carved easily with a knife. Practically all soapstone which has been produced commercially is in reality a metamorphic rock, composed of only 50 to 80 per cent of talc (hydrous metasilicate of magnesium) intimately mixed with varying proportions of chlorite, amphibole, pyroxene, and mica, as well as pyrite, quartz, calcite, or dolomite. Soapstone, therefore, generally is a rock rather than a mineral, although the term may be used accurately to refer to massive talc, however pure. Regardless of the indefinite mineral composition of the material, however, analyses of typical samples reveal that the magnesia content of commercial-grade soapstone is at least 20 per cent. A parallel situation exists

with limestone and calcite. A bed of impure limestone containing only 60 to 75 per cent of calcium carbonate can not be called calcite, although a deposit of pure calcite may justly be designated as limestone.

This paper is concerned only with soapstone, and throughout this discussion the term will be used as it properly is applied in present-day commercial nomenclature.

HISTORY

Probably the earliest known uses of soapstone were dependent upon the ease with which it could be cut. Early Egyptian scarabs and amulets were carved steatite coated with a blue vitreous glaze. The Assyrians used it for signets, and numerous carbed steatite ornaments were found among ruins in Rhodesia. For centuries the Chinese and Japanese have carved weird figures out of steatite or agalmatolite; the handiwork of the present generation may be purchased in any Oriental curio shop.

The cathedral of Trondhjem, Norway, was built about 1200 A. D. of soapstone from Gudbransdal. The structure recently was rebuilt to repair damage due to vandalism and fire, but otherwise the stone is in good condition. It has hardened considerably since the building was first erected. Steatite found in association with serpentine near Cornwall, England, was used in the manufacture of old Worcester porcelain.

Soapstone was first used in the United States by the American Indians, who early recognized its heat-retaining qualities. Soapstone bowls, pots, cooking stones, and other utensils made by the Indians are on display in many museums. This utilization gave rise to the term "potstone," which still is applied to soapstone in some localities.

The deposit of soapstone in Albemarle County, Va., was first opened on a semicommercial scale about 1880. From then until now the history of the soapstone industry parallels that of the Virginia Alberene Corporation. John G. Porter was responsible for the development of the deposit, and the present company is a direct outgrowth of his early activities. Soapstone has been produced from this deposit for more than 50 years.

The original quarries were located at Alberene, Va., but present activities are centered at Schuyler, just across the Albemarle County line in Nelson County. Several quarries are operated. The exact number as well as their output is directly dependent upon demand, but the usual average is four or five.

Other States have produced soapstone, but Virginia has dominated the industry. In 1913, there was 1 producer each in Maryland, North Carolina, and Rhode Island; 2 in Vermont; and 6 in Virginia, the latter State producing more than 85 per cent of the total. The following year Virginia produced 15 times as much soapstone as the other four States, and in 1915, 23 times as much. Production in 1916 was virtually all from four quarries in Virginia,

although California and Rhode Island reported activity. The entire output for the following year was from three Virginia quarries. For the next three years Vermont and California reported sporadic production, but in 1921 the Virginia Alberene Corporation was the only producer of any consequence.

From 1922 to 1926 four new companies in Virginia and one in North Carolina were organized to exploit soapstone, but in 1927 only one company remained active. Meager reports from California indicate that in 1923 soapstone from Oroville, Butte County, Calif., was being shipped to chemical plants in the San Francisco district. Later information, however, has intimated that this material may have been talc rather than soapstone. Since 1929, the product of the Virginia corporation has been the only soapstone extensively marketed, although locally small quantities of similar material may have been produced.

COMPOSITION AND PROPERTIES

Soapstone is typical of all metamorphic rocks in that its chemical and physical properties may vary considerably. Since talc is the essential mineral of soapstone, analyses of most samples show some resemblance to the theoretical composition of talc, which is: SiO_2, 63.5 per cent; MgO, 31.7 per cent; and H_2O, 4.8 per cent. Iron oxide, alumina, and lime are the other principal constituents of soapstone. Results of analyses made by Mellon Institute of seven samples of soapstone are summarized in the following table.

Chemical Composition of Soapstone[1]

	Maximum	Minimum	Average
Silica	40.20	34.12	36.11
Alumina	9.10	5.45	6.78
Iron oxide ..	11.75	8.84	9.96
Lime	6.39	3.19	3.99
Magnesia	29.03	23.69	26.48
Titania	0.48	0.28	0.35
Alkalies	0.37	0.13	0.23
Ignition loss	18.36	8.44	14.87

1 - Courtesy of Virginia Alberene Corporation.

Soapstone which contains no carbonate impurities is comparatively inert; chemically even at high temperatures it resists the action of strong acids and alkalies. The durability of the stone depends upon this same property. Although some grades may be easily abraded, soapstone shows little or no deterioration when exposed to weathering agencies, even for very long periods.

Soapstone is a dense, massive, fine-grained rock. It varies in color from light gray through shades of blue to dark blue with occasionally a greenish cast. The average weight of commercial stone is about 180 pounds a

cubic foot, which is equivalent to a specific gravity of nearly 2.9. It is virtually nonabsorbent; in fact, most soapstones show an absorption by weight of less than 0.1 per cent. Locally, the stone may be fractured or may lack uniformity. The deposits, however, have no regular system of joints or cleavage, although the grain of the rock usually is pronounced. Soapstone has little tendency to split or spall.

The hardness of soapstone varies considerably; that now being produced commercially is divided into three grades: soft, regular, and hard. The soft stone is typical of the usual conception of soapstone. It is soft enough to be scratched with a finger nail, but its strength is satisfactory for numerous uses. The hard grade contains a sufficiently high percentage of siliceous material to make it considerably harder and more durable than common soapstone. It is best suited for stair treads, floor tile, and similar uses. The so-called regular grade is by far the most abundant. Its properties are midway between the hard and the soft, and virtually all fabricated soapstone equipment with interlocking joints is made of this quality of stone.

The ease of workability is the most valuable characteristic of soapstone. Its comparative softness and its massive uniformity combine to make it the most easily fabricated of all natural stones. It may be tongued, grooved, trimmed, or bored with ordinary woodworking tools. Regardless of these characteristics, however, soapstone has remarkable transverse and compressive strengths. The following table gives a summary of tests made of representative soapstone samples.

Physical Strength of Soapstone[1]

Thickness of sample,[2] inches		Fibre stress per square inch at rupture, pounds					
		Transverse strength			Compressive strength		
		Soft	Regular	Hard	Soft	Regular	Hard
1	Maximum	2190	2590	4120	5200	7000	14300
	Minimum	1930	2530	3730	3300	6100	8900
	Average	2060	2560	3930	4150	6580	11420
$1\frac{1}{4}$	Maximum	2200	4250	3110	3800		
	Minimum	2050	2550	2630	3500		
	Average	2140	3400	2870	3650		
$1\frac{1}{2}$	Maximum	2390	3000	3670			
	Minimum	2160	2680	3050			
	Average	2310	2840	3370			
2	Maximum	3420	3960	3590	8000	15300	11300
	Minimum	1870	2930	3010	3100	6300	5100
	Average	2640	3450	3300	4650	9880	8530
Average of all samples		2290	3060	3370	4320	8230	9980

1 - Courtesy Virginia Alberene Corporation.

2 - Samples for transverse tests 4 by 14 inches, supported on knife edges, 12-inch span, loaded at center; samples for compressive tests, cubes cut from transverse test samples.

Soapstone takes up heat slowly, but once hot it tends to hold its absorbed heat considerably longer than other natural stones. For all practical purposes it is fireproof, but, upon heating to a temperature of about 1,800° F., it changes color and becomes quite hard. The specific heat and the dielectric strength of soapstone are unusually high. All these properties are utilized in various specialized applications of the stone.

USES

The various uses of soapstone are related to both its chemical and its physical characteristics. Several of the most common uses are dependent upon its comparative chemical inertness, some upon a combination of chemical composition and physical properties, and others have no direct relation to either.

The chemical inertness of soapstone has led to its wide use for laboratory equipment; in fact, many modern laboratories are completely furnished with soapstone table tops, sinks, drain boards, gutters, fume hoods, shelving, floors, and tanks as well as miscellaneous special equipment. Similar use of the stone is made in acid and other chemical plants for vats, floors, shelves, wainscoting, and other parts of the plant which may be subjected to the destructive action of chemicals or fumes. These applications consume only about 10 per cent of the total volume of soapstone produced.

The use of the stone in dye plants depends upon its low absorption as well as its chemical inertness. When soapstone tanks are used, it has been found possible to change from one color to another with no appreciable contamination by merely rinsing the tank thoroughly between solutions. These properties have encouraged its use in storage tanks for acid or alkali solutions. Surface stains are removed easily by "Carborundum" honing or sandpapering.

In addition to the laboratory equipment already mentioned, soapstone is used for a variety of purposes in which the general adaptability of the stone is more important than its specific property of acid or alkali resistance. Among these uses are for photograph, blueprint, X-ray, and cardiagraph developing tanks; autopsy, mortuary, and mausoleum slabs; animal kennels and cages, particularly for hospitals and related institutions; and miscellaneous equipment for all types of research laboratories.

A good description of the use of soapstone in the paper industry is given in "Talc and Soapstone in Canada" by Spence.[3]

> One of the most extensive uses of block soapstone is for lining the smelting furnaces of sulphate (kraft) pulp mills. In such mills, furnaces are used to smelt the black ash formed during the initial stage of recovery of the alkali in the waste liquor from the digesters. These furnaces are of stationary type and are lined with

3 - Spence, H. S., Talc and Soapstone in Canada: Canada Dept. Mines, 1922, p. 55.

soapstone. Into them is introduced the black ash, mixed with sulphate of soda and sawdust, and the whole is then burned, the fused alkali being run out into a dissolving tank.

In addition to its resistance to fire and alkali under such conditions, soapstone has the merit of offering a surface to which the fused alkali will not adhere, and thus the furnace walls do not require to be cleaned. *****

The requirements in a soapstone for sulphate pulp furnaces are: fine to medium grain, compactness and homogeneous composition, and freedom from flaws and cracks. It should consist largely of talc, and contain no carbonates (dolomite, calcite) or pyrites. The stone should possess a massive, as opposed to a schistose texture, since schistose soapstone tends to spall readily and has little strength.

The discovery of a soapstone possessing the above characteristics, in Canada, would be of considerable benefit to domestic paper mills, since the quantity used is large and the cost of the imported stone high - from $5 to $6 per cubic foot, laid down.

The soapstone bricks used vary in size. Common dimensions are: 12 x 12 x 6 inches; 12 x 6 x 6 inches; 12 x 6 x 3 inches; 18 x 12 x 8 inches; 18 x 12 x 12 inches.

Before 1910 virtually all soapstone used in American paper mills was imported from Sweden. The American mills were gradually learning the advantages of dealing with an American source of supply when the World War brought on chaotic trade conditions which accelerated the change already taking place. Actual use of stone from the Virginia deposits showed that it was satisfactory in quality and that the service from domestic sources was superior to that from imports. Consequently, this use has increased until it now comprises about 40 per cent of the total output.

The other common uses of soapstone are more dependent upon its physical properties than its chemical composition. Laundry tubs and kitchen sinks are the oldest representatives of this group. They were first made at Alberene, Va., about 60 years ago. For more than 50 years, they represented the greater part of the soapstone business. Within the past 10 years, however, the laundry tub market has decreased until now it comprises only about 10 per cent of the total volume of soapstone production. Competitive materials, electric washing machines, partial saturation of potential markets, promotional activities of laundries, and the trend toward apartments have been responsible for the slump in the demand for laundry tubs. The sale of tubs finished in bright colors has aided in meeting the desire for color in home furnishings.

The durability, uniformity, workability, comparative nonabsorption, and neutral color of soapstone make it desirable for numerous architectural adaptations. The increased use of soapstone in building construction during the past few years has more than offset the decline in laundry tub production.

Large quantities of soapstone have beenused for sanitary partitions, shower compartments, sills, interior trim, wainscoting, spandrels, and exterior trim. Two other important architectural uses, stair treads and floor tile, are the principal outlets for the hard grades. The use of soapstone in combination with marble and other decorative stone is being actively promoted at present. The greatest possibility for expansion of soapstone production is generally acknowledged to be through the increased use of the stone for various architectural purposes. In fact, at present soapstone logically should be included in any discussion of the dimension stone industry. The volume of production is small when compared with other more common building stones, but in its particular field, soapstone is of noticeable importance.

The high dielectric strength of soapstone, combined with its uniform texture, low absorption, freedom from metallic veins, resistance to flame action and high mechanical strength, makes it an especially desirable material for electrical insulation. Here again, however, the advantage of easy workability must not be underestimated, for the rapidity and accuracy with which soapstone can be machined undoubtedly is directly responsible for much of its popularity among electrical manufacturers. It is used for switchboards, panels, barriers, bases, circuit breaker compartments, insulating floor slabs, battery room shelving or flooring, and for other related purposes.

The high heat-retention properties of soapstone led to its use as potstones and cooking utensils by both the Indians and the early American settlers. Until the last few years an appreciable part of soapstone production was utilized in the manufacture of griddles, fireless cookers, oven floors and grates, footwarmers, snow melters, and hearths. At present, however, more modern contrivances have supplanted most of these products although soapstone still is being used in large commercial travelling ovens and electrical fireless cookers.

When heated to a temperature of about 1,800° F., some grades of soapstone turn to a golden brown color and become so hard that they can not be scratched with an ordinary knife. Considerable research work has been done in attempts to utilize this property commercially in the production of floor tile or decorative stone for use where hardness and durability are important factors. Results obtained in using the material have been entirely satisfactory, but manufacture is difficult. The principal constituent minerals of soapstone contain water of crystallization which is driven off at varying temperatures, depending upon the exact composition. In order to avoid cracking while the soapstone is being heated, the temperature must be held constant for several minutes at the critical point or points of the particular sample. Obviously, in mass production with varying critical temperatures this is impractical and, as a result, in the manufacture of such tile a heavy loss is incurred because of fracturing.

A modification of this same principle which apparently has possibilities is heating of soapstone dust, the resultant product to be used as an abrasive. Various grades and sizes could be produced. The process is patented, but no commercial utilization has been attempted.

Only an insignificant part of soapstone waste is now being utilized profitably. Small quantities of irregularly broken slabs are used locally in landscaping as stepping stones or for rock gardens. Unfortunately, the isolated location of the deposits prevents extensive development of this market possibility. Soapstone has been used in Schuyler and vicinity for masonry construction. Crushed soapstone has been used satisfactorily as concrete aggregate, but this outlet logically is confined to local utilization.

The only outlet for waste soapstone of any appreciable size or importance is in pulverized form. Ground soapstone has been used for a variety of purposes in direct competition with other pulverized rock including the lower grades of talc. It does not compete with high-grade talc, especially for uses in which color is an important factor. At present the rubber and roofing trades are the largest consumers. It is used principally for dusting raw or processed material in various stages of manufacture to prevent sticking, although it is reported that some of it may be used as a filler. Considerable quantities are exported; one large rubber company in Singapore using it for dusting raw rubber to be shipped into this country. Much of the tire powder used to sprinkle between casings and inner tubes is ground soapstone.

Ground soapstone has been found to be satisfactory for rock dusting of coal mines, a precautionary measure extensively used in the coal mining industry to prevent the spread and the occurrence of coal-dust explosions. However, in most mining districts competition from low-priced ground limestone is severe. The extremely low absorption of soapstone should be desirable in connection with rock dusting, but no studies have brought out this feature.

The use of admixtures in concrete has attracted much attention during the past few years. Numerous products have been introduced, all of which claim increased workability and denser concrete as their principal advantages. Ground soapstone has been used successfully for this purpose, but no attempts have been made to promote its distribution on a large scale.

The manufacture of floor tile made of ground soapstone, sawdust, and Portland cement, appears to be worthy of further research. This type of flooring combines resilience and durability, and it is especially adaptable to industrial buildings. Its commercial development no doubt would stimulate production of ground soapstone.

GEOLOGY

Although numerous authorities have studied the geology of soapstone, it is generally agreed that much additional work must be done to determine its origin in detail. The consensus, however, is that soapstone is secondary, having been formed usually by the alteration of a basic igneous rock. Such rocks necessarily would have to be high in magnesia; as for example, pyroxenite, amphibolite, dunite, or peridotite. In some minor occurrences, soapstone has resulted from the metamorphosis of dolomitic limestone. Most deposits characteristically occur in long narrow belts roughly parallel to the general trend

of the country rock. This orientation suggests derivation from sedimentary rocks, but additional evidence in many localities virtually proves that the soapstone bodies are dikes rather than strata.

The soapstone deposits near Schuyler, Va., occur as lenses and irregularly shaped, elongated bodies lying in or along the edges of larger intrusions of partially metamorphosed gabbro or pyroxenite. Cambrian schists predominate throughout the region. The general strike of the schists is from N. 40° E. to N. 60° E., and the dip varies from 50 to 80° to the southeast. Both the meta-igneous intrusions and the soapstone dikes lie roughly parallel to the strike of the country rock, although locally this orientation may vary. The soapstone lenses range in size from virtually nothing up to immense bodies 1,500 to 2,000 feet long and 300 feet wide. Their depth is not known. At the present rate of production, there is enough available soapstone at and near Schuyler to last for 50 to 100 years.

Burfoot[4] has made a comprehensive study of the origin of the talc and soapstone deposits of Virginia. His complete report ultimately will be available as a bulletin of the Virginia Geological Survey. In an interesting paper originally presented at the 1929 annual meeting of the A.I.M.E., Ryan[5] discusses the geology of Virginia soapstone, especially with reference to mining problems. These references should be consulted for additional information on the origin and geology of soapstone.

GEOGRAPHICAL DISTRIBUTION

The existing confusion in the use of terms "talc," "steatite," and "soapstone" makes it extremely difficult to correlate reports of deposits with soapstone as it commonly is known. In referring to some foreign deposits, the terms apparently are used interchangeably.

Virginia has virtually a world monopoly on the production of slab and block soapstone. The principal Virginia deposits form a large belt which extends through Nelson, Albemarle, and Orange Counties. At present, all operations are centered at Schuyler, which is in Nelson County only a short distance from the Albemarle line. Minor deposits of soapstone occurring in association with talc have been worked in Fairfax, Franklin, Amelia, and Henry Counties.

Soapstone has been produced in Vermont, Maryland, North Carolina, Rhode Island, and New Hampshire, but all these operations have long since been abandoned. Little is known regarding any of these deposits, but cessation of activity apparently was caused in most instances by a combination of technical and economic difficulties. In 1930, soapstone production was reported in California, but available information regarding it indicates that it more

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4 - Burfoot, J. D., jr., The Origin of the Talc and Soapstone Deposits of Virginia: Jour. Econ. Geol., vol. 25, 1930, pp. 805-826.

5 - Ryan, C. W., Soapstone Mining in Virginia: Am. Inst. Min. and Met. Eng., Tech. Pub. 160, 1929, 31 pp.

properly should be classified as talc. A large deposit of soapstone reported near Benton, Arkansas, never has attracted much attention because of its inaccessible location.

The Canadian paper manufacturing industry constitutes an important market for block soapstone, much of which now is supplied from the United States. Since about 1924, however, considerable interest has been shown in the Canadian deposits, and production has been reported from the Lake of the Woods region of western Ontario and from the Province of Quebec. These deposits are located relatively close to many of the large paper mills, and it is reasonable to expect increased production, possibly to the ultimate exclusion of Virginia stone.

Deposits of soapstone in the Scandinavian countries have been worked for many years, principally for furnace blocks and structural purposes. Numerous occurrences of soapstone have been reported in India, China, and other countries, but none has been developed commercially except as a local source of foot-warmers, carved ornaments, grates, or similar minor commodities.

PROSPECTING

No development work in any soapstone deposit ever should be attempted without complete knowledge of its extent, quality, and uniformity. A careful scrutiny of outcrops will reveal something of the nature of the stone and the value of such a reconnaissance is considerable, but additional information generally is necessary to determine the actual worth of the deposit. Ryan's[6] opinion is worth consideration.

> The study of surface indications can give at best only approximate ideas as to the nature of the underlying stone. The best indicator is the outcrop of uniform resistance to the elements, with an even distribution of talc and a relative freedom from carbonate impurities. The core drill must be relied upon to verify or nullify these indications.
>
> Formerly a series of trenches was dug at promising locations to uncover the outcrop more completely, as it was thought that a more satisfactory study of the qualities of the stone could be made in that way. This method is expensive and can not compare with core-drilling, either in cost or scope of information obtained. The diamond drill will show the conditions at depth, will indicate the proportions and nature of the various grades of stone, and will give some information of fissures.

The value of consulting the State geologist should not be overlooked. Many States maintain a well-equipped, competent geological survey, the members of which may have accurate geologic knowledge of the deposits in question. Ordinarily this service is free, and the information received may prevent unwise expenditures both in prospecting and development.

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6 - Ryan, C. W., Work cited, p. 17.

If a preliminary investigation appears to justify the expense of developing the deposit, a competent engineer should be employed. His duties should be twofold; first, to check the results of the original prospectors and enlarge on that information; and second, to supervise the actual development. The importance of this procedure can not be overestimated.

QUARRYING

The limited scope of this report prohibits a detailed discussion of soapstone quarrying methods. However, the problems are virtually identical to those encountered in marble quarrying, except that soapstone is softer and no attention has to be given to the position of markings like those which enhance the value of polished marbles. On the other hand, the softness of soapstone necessitates extreme care in handling it. An excellent discussion of all phases of quarrying and preparation of marble is given in a paper by Bowles.[7] This report no longer is available for distribution, but it can be consulted in most libraries.

The paper by Ryan[8] is devoted directly to the mining of soapstone. Numerous other articles of lesser importance on various phases of the industry have been published in trade journals. The following brief discussion describes practices at Schuyler, Va., the home of the only active producing company.

The normal size of quarries at Schuyler is 100 feet long by 100 to 120 feet wide. The actual width is governed by the size of the dike. Sufficient soapstone to provide a good face is left in place along the hanging wall. When a series of quarries is opened along the dike, pillars 22 feet wide are left standing between operations. The top five or six floors of these pillars may be salvaged, but usually they are left standing to support the walls of the abandoned quarries. The ultimate depth of any quarry depends directly upon the condition of the walls. Some operations have been abandoned when only about 100 feet deep, but the average depth is nearly 200 feet. The deepest quarry ever operated at Schuyler was 271 feet.

The soapstone operator should have a thorough understanding of wall conditions, for the safety of the men working in the pit depends directly upon him. Quarry walls like mine roofs rarely give way without showing preliminary evidences of weakness. A superintendent with mining experience is an asset in soapstone quarrying for sound knowledge of conditions in the pit not only may avoid serious accidents, but also may permit deeper working of quarries which under less competent management would be abandoned at some arbitrary depth.

The method as well as the expense of opening a new quarry depends directly upon local geologic conditions. Overburden generally is removed by steam

7 - Bowles, Oliver, The Technology of Marble Quarrying: Bull. 106, Bureau of Mines, 1916, 174 pp.

8 - Ryan, C. W., Work cited.

shovels and drag scrapers. No explosives are used during either development or quarrying. Hydraulic stripping has been used in some instances but not extensively. Occasionally, conditions are such that good stone is found on the top floor, thereby greatly reducing the quantity of waste stone to be handled. More commonly, however, the first seven or eight floors are principally waste.

Disposal of overburden and waste rock is an important problem in the soapstone industry. Dumps near operating quarries tend to aggravate the stress upon the quarry walls. It is desirable, therefore, that waste material be dumped where it will not interfere with present or future operations. The most satisfactory method is to fill abandoned pits with the refuse from newly opened quarries. Mill waste also must be considered, for only about 20 per cent of the quarry output emerges from the mill as finished product. Under favorable conditions, this material may be disposed of economically by dumping it into nearby abandoned quarries.

After the overburden is removed, quarry operations are commenced by channeling across the strike with either steam or electric-air channeling machines. The distance between channel cuts varies from 4 to 6 feet and the average depth is 6½ feet. The center cut is broken out first, and thereafter all channel cuts are undercut to their full width. The undercutter is a reciprocating machine which works on exactly the same principal as the channeler. It makes a cut slightly inclined from the horizontal. A Jeffrey longwall undercutter equipped with Stellite teeth is being used satisfactorily for undercutting in quarries producing "soft" stone.

When the entire width of the quarry has been channeled and undercut, one end of the section is channeled and the end block is broken out. All blocks except the end ones are roughly diamond shaped because all breaks are induced along the natural grain of the rock, the dip of which may vary from 30 to 60°. The blocks are broken out by drilling a series of holes along the natural parting planes of the rock and then splitting by wedging. The size of the resultant diamond-shaped blocks is roughly 4 by 4 by 6 feet or slightly larger. Irregularly shaped or considerably larger blocks may be quarried for special purposes. Each block is graded according to hardness, color, and soundness. The uses for every block depend upon its grade. Swinging boom derricks are used to lift the blocks from the quarry floor and to place them on cars or stock piles, depending upon the current mill requirements.

MILLING

The first step in the preparation of soapstone is gang sawing of the blocks. The spacing of cuts depends upon the uses for which each block is intended. A high percentage of the material is sawed into thin slabs, therefore, the diamond shape of the blocks is not conducive to excess waste as would be the case if large square or rectangular blocks were the ultimate product. Furthermore, for most uses the stone must be sawed parallel to the rift.

There are 38 gangs at Schuyler, the actual number in operation at any time depending upon the mill output. The saws are semiautomatic, one man operates

a battery of four. Sea sand, principally from Norfolk, Virginia, is the abrasive commonly used; minus 30 plus 46 mesh has been found to be most satisfactory. The sand is recovered through a trap at the bottom of each gang and used over and over, fresh sand being added at the discretion of the operator. In normal times 5 or 6 cubic yards of sand a day is sufficient to supply all saws.

Each blade of a gang is a steel strip $3\frac{1}{2}$ inches wide by 12 feet long and 1/8 inch thick. This length blade with an 18 inch stroke can be used to saw a block up to 10 feet in length. The average speed of sawing during the day is 84 complete strokes a minute, but at night when there is no power load at the mill the speed steps up to about 100 strokes. The rate of cutting varies considerably with the grade of the stone, but it averages about 4 linear inches an hour. In normal practice it takes a little more than one 12-hour shift to saw a 65-inch block.

Slabs are taken from the gang saws either to the stock mill or to the custom mill. The cut blocks are handled by an overhead crane, while in the mill individual slabs are conveyed by a monorail system. The stock mill produces routine, standardized products such as laundry tubs, sinks, and furnace blocks.

Slabs to be used for laundry tubs are trimmed with a steel-toothed band saw similar to those used in woodworking. Laundry tub slabs are gang-sawed 1/8 inch thicker than the finished product. Therefore, after trimming, each slab is taken to a rubbing bed where the excess thickness is removed and a rubbed finish is imparted to the stone. A cast-steel bed 4 inches thick and 46-mesh crushed steel abrasive are the principal features of this equipment. The life of each bed is from 2 to 3 years. By means of an eccentric the stone is moved back and forth across the bed in order to prevent grooving and to insure uniform wear on the bed itself. After having been tongued and grooved by "Carborundum" machines and smoothed by "Carborundum" finishers, the slabs are ready for the assemblers.

One small bolt in each corner is used in assembling laundry tubs, but no part of it is exposed in the interior. Putty made of linseed oil, litharge, and a small percentage of whiting binds the tubs together. This putty expands as it seasons, thereby insuring water-tight joints. In summer it requires only 6 to 10 days to set properly, but in cold, damp weather it may take 2 months for its final set. Plaster of Paris is used to relieve the strain on the bottoms of the tubs while the putty is seasoning.

Soapstone, like marble and other ornamental stones, must be handled carefully. Fortunately, however, small fractures can be repaired so effectively that the break is unnoticeable. German marble cement is used to patch broken corners or similar fractures. The slab is rubbed smooth at the break, and a piece of soapstone matching it in color and texture is smoothed to fit the surface of the slab. They are cemented, and the repair piece is honed down to conform with the shape of the original slab. The cement ordinarily is stronger than the stone itself, and the repair can be detected only by minute observation

when the stone is wet. A patching cement made by mixing soapstone dust with marble cement is used to patch small breaks or to fill up holes which have been gouged into the stone. This cement is heated and pressed into place. When cool, the roughness is sanded off and the repair is found to blend perfectly with the stone. A few of the old-timers in the mill are experts at mixing and using this patching material.

A very important function of the stock mill is the production of furnace blocks for use in paper manufacturing plants. Numerous sizes and shapes varying from 3 inches to 3 feet are produced. All cutting is done by a circular diamond saw. The most important consideration in the manufacture of furnace blocks is the grain of the stone. Blocks laid in the furnace with the grain parallel to the exposed surface would spall, thereby giving unsatisfactory service.

At present all other soapstone equipment is fabricated to specifications in the custom mill. The general procedure is similar to the stock mill except that blueprints are followed on all jobs. Furthermore, much stone used for the products of the custom mill is harder than that used in laundry tubs; as a result, "Carborundum" circular saws rather than steel are used for trimming. "Carborundum" grinders supplement rubbing beds; usually one side is levelled on a rubbing bed after which the slab is ground to the proper thickness on the "Carborundum" machine. Each slab then passes a checker who designates from blueprints the additional fabricating work to be done. Joiners, drill presses, and similar equipment are used to complete the preparation of each piece. The actual assembling of the slabs is done either in the mill or on the job, depending upon the nature of the order. In either event, the linseed oil and litharge cement commonly is used. Throughout the entire procedure similarity to woodworking methods is strikingly evident. This, of course, is due to the nature of the stone and the ease with which it may be fabricated.

A small part of the better-grade waste soapstone is ground in a mill at Damon, Va., about 4 miles from Schuyler. This mill has been designed for flexibility of product. A gyratory crusher reduces the stone to 1 inch. It is then screened, the fines passing directly to an Allis-Chalmers ball mill while the coarse split goes through a Pennsylvania hammer mill, which reduces it to about 1/4 to 3/16 inches, before passing to the ball mill. Hum-mer screens are used for the next sizing, the mesh depending upon the product desired. Oversize goes through an Allis-Chalmers tube mill, and the fines from both are subjected to air classification to collect the ultimate product, 97 per cent of which passes 300 mesh. The most important product of the mill is minus 40 plus 48 mesh, which is used extensively in the rubber industry.

MARKETS AND PRICES

Soapstone is sold by the producer (through a selling company) direct to the consumer. There are virtually no merchants, middlemen, or independent fabricating plants. Small quantities of rough blocks have been marketed sporadically in past years, but the outlet for raw material is inconsequential. Markets for soapstone are world wide, but only a small proportion of present production is exported. The largest consumption in the United States is east

of the Mississippi within reasonable shipping distances of Schuyler. In recent years the increased use of soapstone for architectural purposes has resulted in a seasonal fluctuation in demand paralleling that of the building industry.

Schuyler is somewhat isolated and therefore all soapstone is shipped by rail. Whenever practical it is shipped in carload lots; the minimum carload being 24,000 to 36,000 pounds. Most soapstone, except furnace blocks, is transported in strong crates to aid in eliminating the hazards of breakage in transit. Slab stone commonly is shipped on edge, for the possibility of breakage is less than when placed flat on the bottom of the car. Few shipments are insured.

The fact that the production and sale of soapstone is in the hands of a single company does not mean that competition is nonexistent. Actually, soapstone meets strong competition from other products in virtually every field.

The prices of soapstone are directly dependent upon the sales prices of competitive materials as well as upon the cost of quarrying and fabrication. Crude soapstone is worth only slightly more than the cost of production; therefore an undeveloped deposit has little value unless it is so located that it would have decidedly advantageous freight rates to marketing centers.

The present unit of measurement for manufactured soapstone is 1 square foot, 1-1/4 inches thick. All products, regardless of their size, shape, or use, are reduced to these units. Furnace blocks comprise the largest low-priced outlet, while complicated developing tanks and similar equipment bring the highest prices. The present range is from $0.30 to $1 a square foot. This is equivalent to $30 to $100 a ton.

PRODUCTION

Figures for soapstone production since 1924 are not available for publication. The accompanying table, however, gives a summary of production from 1916 to 1924.

Soapstone sold by producers in the United States, 1916-1924[1]

Year	Short tons	Value
1916	19,127	$ 489,606
1917	19,885	402,506
1918	12,330	501,059
1919	16,504	530,163
1920	19,707	709,400
1921	17,423[2]	627,826[2]
1922	22,700	712,144
1923	22,857	932,098
1924[3]	25,630	1,288,885

1 - Compiled from Mineral Resources of the United States.

2 - Includes sawed and manufactured talc.

3 - Figures since 1924 not available.

A summary of the relation between output and uses is given in the accompanying table. It is interesting to note that the two principal outlets for soapstone in 1930 were of little importance in the early years of development of the industry.

Principal outlets for soapstone showing proportionate production, 1900-1930, per cent

Use	1900	1910	1920	1930
Laundry tubs (includes sinks)	85	75	59	10
Furnace blocks	0	0	4	40
Special (includes architectural)	15	25	37	50
Miscellaneous	-	-	-	Less than 1

IMPORTS AND EXPORTS

With the exception of Swedish furnace blocks and Chinese carvings, virtually no soapstone ever has been imported, and exports have been confined principally to shipment of furnace blocks to Canada. Foreign production is in no way comparable to the domestic output. Small quantities of soapstone have been produced in Great Britain, Norway, and Sweden. France, Spain, and Germany are reported as having produced soapstone or steatite; but the figures are included with talc, and apparently the soapstone is of little consequence. Canada is now producing soapstone principally in the form of furnace blocks. As yet, however, the Canadian output apparently is not sufficient to meet the needs of the extensive paper manufacturing industry, for considerable quantities of furnace blocks are shipped into Canada from Schuyler.

TARIFF HISTORY

Under various tariff acts soapstone has been dutiable, if imported, at the same rates as talc. The present duties are listed in paragraph 209 of the tariff act of 1930, which reads as follows:

> Talc, steatite or soapstone, and Franch chalk, crude and unground, one-fourth of 1 cent per pounds; ground, washed, powdered, or pulverized (except toilet preparations), 35 per centum ad valorem; cut or sawed, or in blanks, crayons, cubes, disks, or other forms, 1 cent per pound ; manufactures (except toilet preparations), of which talc, steatite or soapstone, or French chalk is the component material of chief value, wholly or partly finished, and not specially provided for, if not decorated, 35 per centum ad valorem, if decorated, 45 per centum ad valorem.

ECONOMIC CONSIDERATIONS

The history of the soapstone industry in the United States is that only one operator has been able to survive. Others have entered the field, but one by one they have dropped out. A survey of the situation reveals several factors which may have been responsible, at least in part, for the failure of former producers. The value of any deposit depends not only upon the quality of the available stone, but also upon its location and extent. Competition from other materials is felt keenly in the marketing of soapstone; slate, marble, concrete, and ceramic materials are the principal competitive products. At best, the total annual volume of material marketed is small; and since it is virtually all in the form of fabricated products, a completely integrated plant comprising extensive milling equipment is needed as well as the usual quarrying machinery. This involves a heavy investment. In the case of a new operation, an even larger expenditure may be required before consumers can be induced to accept a new product. Under these circumstances it should be evident that mere possession of even a good deposit does not in itself indicate that it will be profitable for a newcomer to enter the soapstone industry.

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

CONSUMPTION OF PRIMARY TIN IN THE UNITED STATES DURING 1930

BY

JOHN B. UMHAU

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January, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

CONSUMPTION OF PRIMARY TIN IN THE UNITED STATES DURING 1930[1]

By John B. Umhau[2]

In 1930, 65,448 long tons of virgin or primary tin were consumed by 1,292 establishments in the United States, representing about 91 per cent of the total consumption as computed from imports, exports, and stocks at the beginning and the end of the year. The reported consumption was equivalent to 38 per cent of the world's tin production in 1930, estimated at 171,000 tons, and was 12 per cent less than the 74,369 tons consumed by 1,307 establishments in 1928.

Decreased consumption of new tin in 1930 as compared to 1928 was a reflection of the general industrial depression. All of the tin-consuming industries showed substantial decreases except tin plate and terneplate, white metal, and tinning, which showed small tonnage increases aggregating only 1,200 tons. The largest tonnage decrease was in the use of new tin in babbitt which declined 33 per cent in 1930 as compared to 1928. This was largely attributable to the decreased production of automobiles and trucks, which was 24 per cent lower in 1930 than in 1928. Percentage decreases in other important tin-consuming industries were as follows: Solder 18 per cent, bronze 19 per cent, tin foil 40 per cent, chemicals 23 per cent, tin oxide 44 per cent, and type metal 46 per cent. The distribution of the amount consumed by various uses during 1927, 1928, and 1930 is shown in table 1.

Consumption studies indicate the fact that variation in tin consumption from year to year parallels the fluctuations in demand for the already well-established tin products. For example, the manufacture of motor cars and trucks[3] required only 11,000 tons of tin in 1930 compared to 19,000 tons in 1928; the canning industry continues to require the greater portion of the tin plate and much of the solder. Recently developed industries, while affecting total consumption of bearing metals, solders, etc., to some extent, have made little change in the relative importance of these commodities as consumers of tin.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6564."

2 Common metals division, economics branch, U. S. Bureau of Mines.

3 Facts and Figures of the Automobile Industry: National Automobile Chamber of Commerce, U. S. A., 1931 ed., p. 83 and 1929 ed. p. 13.

Table 1.- Consumption of primary tin in the United States, by uses, 1927, 1928, and 1930

Uses	1927		1928 1/		1930	
	Long tons	Per cent	Long tons	Per cent	Long tons	Per cent
Tin plate and terneplate...	24,525	35.96	27,053	36.38	27,753	42.40
Solder	13,602	19.94	13,874	18.66	11,407	17.43
Babbitt	7,595	11.14	8,150	10.96	5,438	8.31
Bronze	4,664	6.85	4,324	5.81	3,499	5.35
Foil	4,193	6.15	5,068	6.81	3,061	4.68
Collapsible tubes	2,710	3.97	2,864	3.85	3,826 2/	5.84
Chemicals	2,621	3.84	4,246	5.71	3,268	4.99
Tin oxide	1,311	1.93	1,183	1.59	666	1.02
Tinning, brass, copper, tubes, sheets or shells, wire, nails, etc.	2,661	3.90	2,636	3.54	2,814	4.30
White metal	849	1.24	802	1.08	1,117	1.71
Type metal	450	.66	411	.55	223	.34
Castings	1,011	1.48	730	.98	74 3/	.11
Other alloys	556	.81	629	.85	306	.47
Miscellaneous	1,450	2.13	2,399	3.23	1,996	3.05
Total	68,198	100.00	74,369	100.00	65,448	100.00

1/ Revised figures.
2/ Not comparable to prior years owing to the addition of several consumers whose consumption was not included previously.
3/ Pure tin castings only; in previous years some castings of tin alloys were reported under this heading.

The great northeastern and north-central manufacturing belt of the United States in which 75 per cent of the total value of all the products of domestic manufactures is produced, accounts for 97 per cent of the total domestic tin consumption; New York, Pennsylvania, Illinois, Ohio, and Michigan, in this territory, consume over 75 per cent of the total.

Of the 1,292 concerns reporting tin consumption in 1930 only 244 consumed more than 15 long tons (approximately one minimum carload) during the year. Of the balance, 172 individual concerns consumed between 5 and 15 tons each during the year, and the remainder, about two-thirds of the entire number, consumed less than 5 tons in the year. These 1,292 concerns include substantially all the initial tin fabricators through whose plants tin first enters industry; manufacturers who purchase alloys or other tin products for further fabrication were not included.

TIN PLATE AND TERNEPLATE

The consumption of primary tin for tin plate and terneplate manufacture represents the principal single use for tin in this country, and in 1930 accounted for 27,753 long tons, or 43 per cent of the total consumption, as compared to the 27,053 long tons, or 36 per cent in 1928. This increase, amounting to about 2.6 per cent, is due primarily to increase in the food-packing industry in which the greater portion of the tin plate production is used. This industry was not affected by 1930 drought conditions, as the principal canning crops compared favorably with previous years. Moreover, there was increased demand for tin plate for cans to replenish canner's stocks which were depleted by the large 1929 crop. While some new uses for cans were developed in 1930, there was some falling off in exports, and domestic demand for metal containers for paints, varnish, chemicals, oils, etc. In addition to food containers, tin plate is used in the manufacture of containers for gasoline, oil, tobacco, etc. It is also used in bottle caps, lithographing, decorating, gas meters, lanterns, and kitchenware. Terneplate is used for automobile tanks, fire doors, metal furniture, roofing, and in other articles.

Refrigerated food is being marketed in increasing quantities and is supplying the public with food which otherwise would have been marketed in tin-plated containers. Glass and paper containers are replacing tin cans in the packing of many foods and household supplies, and the attractiveness and utility of these containers are bound to expand their popularity. Nonmetallic products, as well as copper, zinc, and zinc-coated roofing have undoubtedly supplanted much tin plate and terneplate used for roofing, siding, etc.

The new continuous sheet-rolling process for manufacturing black plate, which makes possible the production of tin plate in coiled form, presages a radical change for the can-making industry. The coiled product is used in new automatic high-production presses for making can tops and bottoms. Economies are effected by increased speed of operation and decreased production of scrap. In addition it is expected that tin plate in the form of coils eventually will be made at a lower cost than tin-plate sheets.[4]

SOLDER

Solder, next in importance, required 11,407 long tons of tin, or 17.43 per cent of the total consumption in 1930; compared with the 1928 figures, 13,874 long tons and 18.6 per cent, this was a decrease. The development of "lap-welding" and die stamping tin cans has lessened the demand for solder in canning operations and consequently for much tin. The practice of "wiping" joints with solder, now virtually abandoned on all new construction, has also effected a diminishing demand for solder and hence for tin. Welding, brazing, and silver solders are growing competitors of tin solders. For electrical work, where resistance to high temperatures and vibration is required, silver solders and brazing are replacing some tin solders. Cadmium is also proving to be a satisfactory substitute for tin in some solders. "Amaloy," a recently introduced solder containing 98 per cent of lead and only 1 per cent or less of tin, is also curtailing the use of tin in solder.

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4 Crane, E. V., Progress in Pressed Metal: Iron Age, vol. 125, No. 1, Jan. 2, 1930, p. 40.

It is of interest that 25 per cent of the concerns using tin for solders also use tin for making babbitt and bronze.

BABBITT AND BRONZE

The manufacture of babbitt required 5,438 long tons or 8.31 per cent of the total for 1930. This was approximately 33 per cent less than the 8,150 long tons consumed in 1928. The consumption of tin in the manufacture of bronze decreased 19 per cent in 1930, the amounts consumed being 3,499 long tons or 5.35 per cent of the total in 1930, and 4,323 or 6 per cent of the total in 1928.

Babbitt and bronze bearings are now being replaced by roller and ball bearings. It is notable that within the past year the first locomotive ever constructed in this country completely equipped through-out with antifriction-type roller bearings was placed in service.[5] The production of babbitt and bronze was adversely affected by the marked decrease in the production of automobiles and trucks in 1930.

FOIL AND COLLAPSIBLE TUBES

The use of tin in tinfoil decreased from 5,068 tons in 1928 to 3,061 tons in 1930, a decline of 40 per cent. In 1930 the use of tin in foil amounted to less than 5 per cent of the total tin consumption, whereas in 1928 it was nearly 7 per cent. The use of tin in collapsible tubes in 1930 amounted to nearly 6 per cent of the total. The 1930 figure includes tin consumed by 5 manufacturers whose consumption is not included in the 1928 figure; hence the two figures are not comparable. Consumption by the concerns reporting for both years shows a decline in this use of tin.

In the collapsible tube and foil industry aluminum has become a potent competitor of tin, and it has been reported recently that nearly pure zinc, refined electrolytically, shows pronounced ductility and has been used for making collapsible tubes and foil. A transparent cellulose produce "cellophane," which may be had in various colors for wrapping confections, etc., has recently appeared as a substitute for foil.

CHEMICALS

There was a decided decrease in the use of tin for the manufacture of tin oxide and other chemicals. One cause for this decrease was the decline in silk consumption in 1930. Considerable tin in the form of "bichloride," "tin salts," "tin crystals," and "stannous chloride" is used in the manufacture of silk fabrics. Tin oxide is largely used as a constituent of the enamel applied to cooking utensils, refrigerators, and sanitary ware. Zirconium oxide, which is cheaper, is being substituted for some tin oxide by many manufacturers; a mixture of 6 parts tin oxide and 4 parts zirconium oxide is used. Japanning of the small cheaper articles, lacquer finishes on refrigerators, and the vitreous china used for sanitary ware instead of vitreous enamels are all factors which tend ultimately to replace considerable tin.

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5 Blast Furnace and Steel Plant, Locomotive Equipped with Roller Bearings; Vol. 18, No. 7, July, 1930, p. 1147.

TINNING

Concerns using tin for tinning accounted for 2,814 long tons of tin in 1930 and 2,636 tons in 1928. Over 90 per cent of the total tin used for tinning in 1930 was consumed as follows:

	Long tons
Tinning wire	1,073
Equipment used in food preparation	706
Tubes, sheets and shells	485
Brass and bronze	103
Iron castings	64
Retinning	55
Connecting rods and babbitting	48
Rivets and pipe threads	8

Smaller quantities are used for harness, steel drums, toys, etc.

Tin amounting to 1,098 long tons in 1928 and 1,265 long tons in 1917 was accounted for as being used for tinning wire. In 1928, only 312 long tons of tin was reported for tinning utensils used in food preparation.

Aluminum cooking utensils have eliminated much tin formerly used in tinning and retinning cooking utensils.

WHITE METAL

In 1930 tin used for white metal amounted to 1,117 long tons, a gain of 39 per cent over the 802 tons used in 1928. Of the 1930 total, 434 tons was reported used in Britannia metal and 123 tons in pewter.

OTHER ALLOYS

Tin used for "other alloys" accounted for only 306 long tons in 1930 as compared with 629 tons in 1928. Included in the total for this item in 1930 are 20 tons used for cable sheathing, which accounted for 37 tons in 1928. Only 10 tons of tin was used in organ-pipe metal in 1930, whereas 21 tons was used in 1928, and 24 tons in 1917. For fusible plugs about 6 tons of tin was used in 1930, compared to 8 tons in 1928. The tin used in the manufacture of special metal products such as bolster metal, dental alloys, aluminum alloys, die metal, shot, phosphor tin, ounce, and needle metal is included in the item "other alloys."

CASTINGS OF PURE TIN

Only 74 long tons of tin were used for castings of pure tin in 1930, whereas in 1928 this item indicated the use of 730 long tons. These figures are not comparable by reason of changes made in the method of collecting data.

Heretofore it was difficult to obtain satisfactory data from the consumers concerning the use of tin for castings and it was thought that the greater portion of the tin reported as being used for castings was actually used as an ingredient of bronze or other alloy castings. The 1930 questionnaire asked specifically "how much tin was used for castings of pure tin."

MISCELLANEOUS USES

Miscellaneous uses accounted for 1,996 long tons or 3 per cent of the total tin consumed in 1930 as compared to over 2,400 tons in 1928. Among the uses included under this heading is zinc coating (galvanizing) which consumed 107 tons in 1930, 133 tons in 1928, and 105 tons in 1917. Pure tin tubing, pipe, sheet, wire, and related articles required 1,075 long tons in 1930; and bottle caps, stoppers, covers, and sprinkler tops, 18 long tons.

Other articles which the consumers reported under this item in 1930 include hollow ware, cutlery and spoons, smokeless powder and small arms ammunition, pattern work, dust or powder, calf muzzles, hammers, diaphrams, and wire rakes. In addition, during 1928 they reported miscellaneous uses as follows:

Tinning mixture, tin bearings.
Rubber industry.
Lining of soda-fountain tanks.
Bars.
Ball-bearing parts.
Wiring and stamping.
Tin ribbon, tin tubing, tin bar.
Valve seats.
Torches
R. R. bond terminals.
Trolley wheels, banjo rings, etc.
Shelving.
Terneplating conductor hooks.
Building up piston heads.
Metallic packing.
Repair work.
Elbows.

CONSUMERS STOCKS OF TIN

Stocks of tin held by domestic consumers increased from 10,606 long tons on January 1, 1930, to 15,500 tons at the end of the year (December 31, 1930), a net increase of 4,894 tons. During 1928 stocks declined from 9,268 tons at the beginning of the year to 8,587 tons at the end of the year, a net draft of 681 tons. The increase in stocks during 1930 was probably due to the desire of manufacturers to take advantage of the low prices prevailing.

Table 2 shows the stocks of tin held and quantity of tin used by manufacturers groupsed according to their principal product.

Table 2.- Tin stocks in hands of consumers manufacturing specified commodities, 1928 and 1930

(Long tons)

Manufacturers of:	Number of manufacturers		Tin used		Stocks			
					1928		1930	
	1928	1930	1928	1930	Jan.1	Dec.31	Jan.1	Dec.31
Tin and terne plate, or tin and terne plate and other products	16	16	28,778	29,320	5,190	4,337	5,809	8,866
Solder, or solder and other products	116	129	17,067	14,679	1,438	1,523	2,135	3,319
Babbitt, or babbitt and other products	96	76	6,000	2,875	266	275	85	244
Bronze, or bronze and other products	537	643	3,522	2,775	331	314	487	610
White metal, type metal, and other alloys; or white metal, type metal, other alloys, and other products	119	104	1,102	1,054	105	66	82	179
Castings, or castings and other products	102	6	1,247	15	115	123	14	13
Foil, or foil and other products	9	6	4,979	3,231	625	616	99	141
Tubes, or tubes and other products	9	15	2,837	3,773	207	225	459	724
Tinned wares and other products	139	162	1,965	2,520	300	345	653	699
Chemicals and other products	15	27	5,413	3,916	475	575	595	509
Miscellaneous	149	108	1,459	1,290	163	135	188	196
For all purposes	1,307	1,292	74,369	65,448	9,215	8,534	10,606	15,500

CONSUMPTION OF SECONDARY TIN

The United States Bureau of Mines has canvassed the tin-consuming industries to obtain data on the amount of secondary tin used in 1928 and 1930, but inquiry has revealed that some consumers had inadvertently reported less than the actual amount of secondary tin consumed in their plants. It has, therefore, been deemed advisable to withhold publication of the 1930 canvass.

I. C. 6565

February, 1932

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

Information Circular

MINING METHODS AND COSTS AT THE BRADEN COPPER CO.'S MINES, SEWELL, CHILE

By

J. S. Webb and T. W. Skinner

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS AT THE BRADEN COPPER CO.'S MINES, SEWELL, CHILE[1]

By J. S. Webb[2] and T. W. Skinner[3]

INTRODUCTION

This is one of a series of papers dealing with mining methods, practice, and costs being published by the United States Bureau of Mines. In this paper a description of the mining practice in vogue at the Braden unit of the Kennecott Copper Corporation is undertaken.

PRODUCTION

At present (1931) the entire production of ore is being obtained from the Teniente mine. The mining method in use is an adaptation of block undercutting which has been a development of the stoping and pillar caving experience obtained at this property. Improvements in mining practice are constantly being made.

GEOLOGY

The Braden Copper Co.'s property is located in the western cordillera of Chile, 30 miles northeast of Rancagua, 50 miles southeast of Santiago, near the head of the Teniente River, and at an elevation of 7,458 to 9,960 feet.

The western cordillera of the Andes forms a sloping dissected plateau. Near the mines the canyons are deeply trenched with long débris slopes and above these tower precipitous cliffs.

The mines are situated on the steep slopes south of the river, the lowest tunnel being at the bottom of the valley and the uppermost 2,500 feet above that level.

The orebodies lie around the periphery of an explosive vent in the form of crescent-shaped deposits, limited on the inside by the tuff contact, where they are of higher tenor.

The upper limit is formed by the bottom of the oxidized zone, 50 to 100 meters below the surface, and the lower limit by the contact with the primary zone.

The explosive eruption which formed the Braden vent intensely fractured the andesite porphyry surrounding its periphery. The width of the zone of most intense shattering is uneven, ranging from 100 meters to 600 meters, the widest portions being on the northeast side of the crater.

After an interval in which the crater became filled with bedded tuffs, mineralizing solutions rising about the periphery of the old vent deposited quartz, tourmaline, biotite, pyrite, and chalcopyrite in the irregular fractures in the andesite porphyry and formed

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6565."

2 One of the consulting engineers, U. S. Bureau of Mines, and mine superintendent, Braden Copper Co.

3 One of the consulting engineers, U. S. Bureau of Mines, and chief mine engineer, Braden Copper Co.

large bodies of mineralized material with a copper content in most places of between 0.50 and 1.50 per cent, though locally slightly richer. This was the original source of the Braden ores, but their enrichment by descending secondary mineralization was necessary to raise their tenor to a commercial grade.

In the orebody pyrite increases and chalcopyrite decreases in relative abundance outward in all directions from the crater and the limit of ore of a profitable grade is generally coincident with the transition from predominant chalcopyrite to predominant pyrite.

The principal primary ore was originally chalcopyrite, but after secondary enrichment the mineralization is divided between chalcopyrite and chalcocite. The development of chalcocite is greatest in the upper part of the enriched zone, and gradually decreases with depth. In the upper part, for instance, the chalcopyrite has in many places been almost completely replaced by chalcocite, with pyrite replaced to a lesser degree. In the lower part of the zone, chalcocite forms thin films along every fracture, however small, but the pyrite is unaffected.

As stated above, the hanging wall limits of the orebody are definitely defined by the tuff contact, but the footwall is purely a commercial limit of workable ore.

METHODS OF PROSPECTING AND EXPLORATION

No systematic exploration was attempted until the property was taken over by Mr. Braden about 1901. Due to the adverse climatic and topographic conditions, he decided that diamond or churn drilling would be impracticable as a method of prospecting, and therefore proceeded with a series of adits, utilizing some of the old workings where they fitted into his scheme. The plan was to drive these adits around the periphery of the crater described, and then crosscut and raise at intervals to determine the lateral and vertical extent of the ore and also the thickness of the lower grade overlying capping.

METHODS OF SAMPLING AND ESTIMATING TONNAGES AND VALUES

Standard channel samples are cut in the sides of the crosscuts, drifts, and raises, each sample representing three meters of working. The assays corresponding to these samples are plotted on assay maps. The ore limits are then laid out on these maps, on the basis of the assays, thus giving the area of commercial ore on each level. The volume between levels (the level interval is ordinarily 50 meters) is then calculated as the frustum of a pyramid. To obtain the grade of this volume, the area on each level is multiplied by its corresponding average assay, thus obtaining an area per cent. These area per cents are then weighted, giving the grade of the ore between the levels. The upper limits of the top level, for instance, are determined by raises and then the volume is calculated from sections, the grade being arrived at by weighting the raise assays with those on the level.

As a method of grade control a grab sample is taken from each active chute in the mine for every 80 tons of ore drawn. Responsible inspectors are employed to observe carefully that no uncommercial ore is pulled from the chutes. They have authority to close immediately any chute showing signs of dilution. These doubtful chutes are promptly sampled on consecutive days, and should the assays obtained show a lower grade than the predetermined cut-off, the chute is definitely abandoned. If these assays show values higher than the required cut-off grade, the chute is then reopened for further draw.

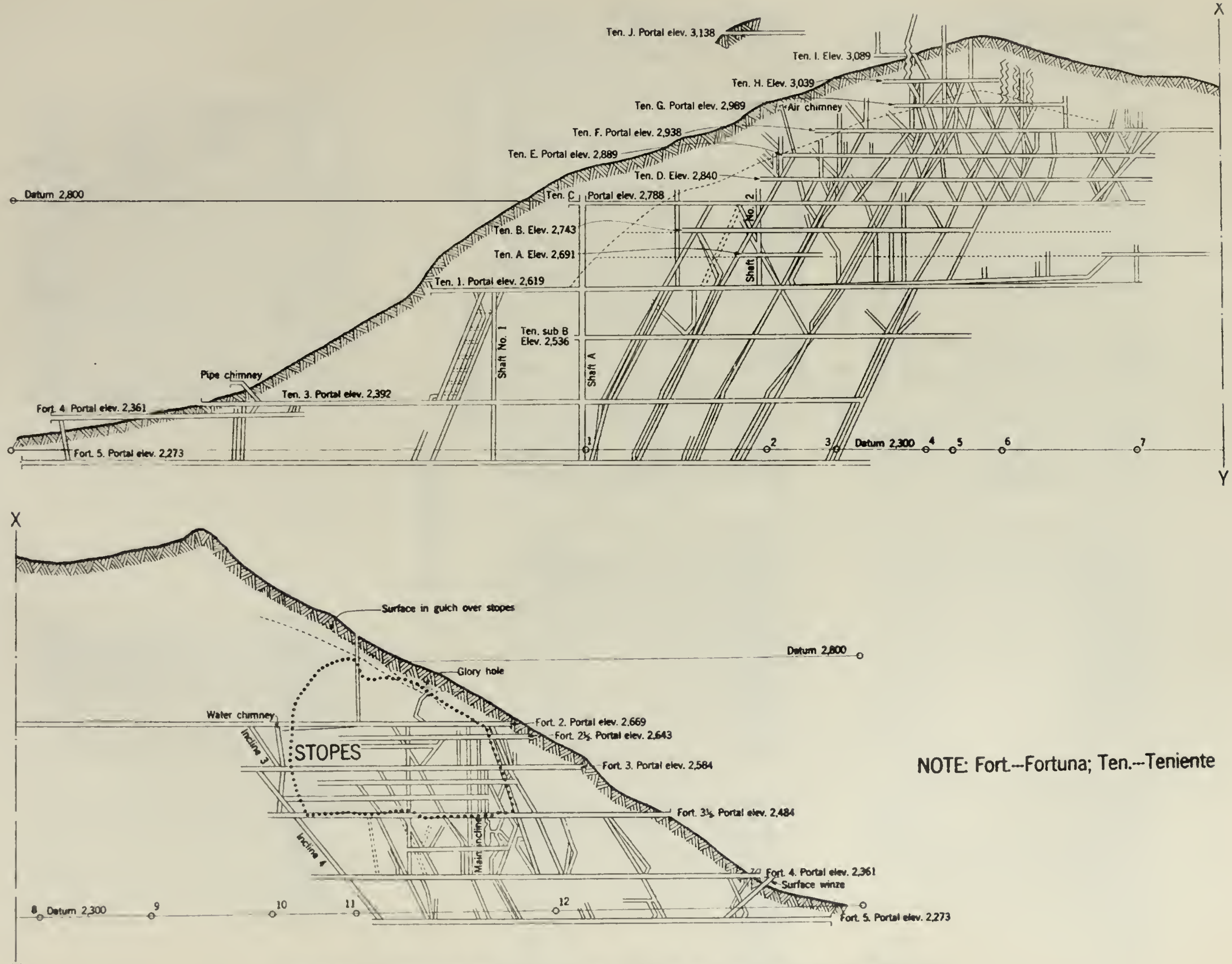
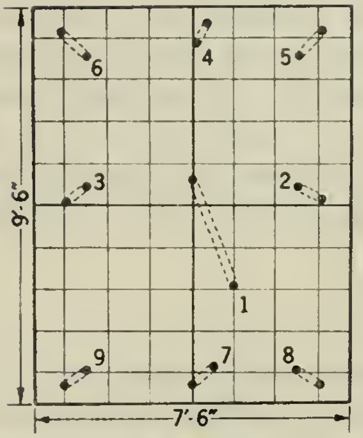
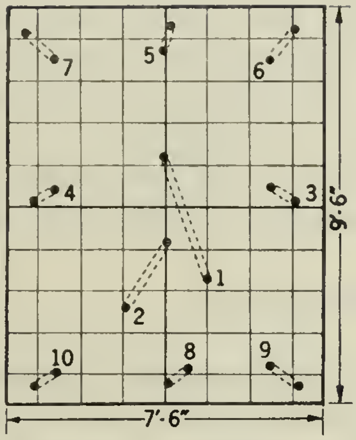


Figure 1.–Longitudinal section of Braden mine, approximately on hanging wall



TYPE A, 9 HOLES

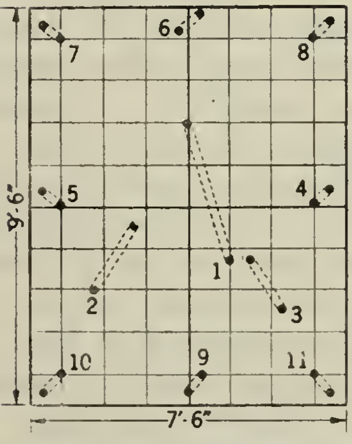


TYPE B, 10 HOLES

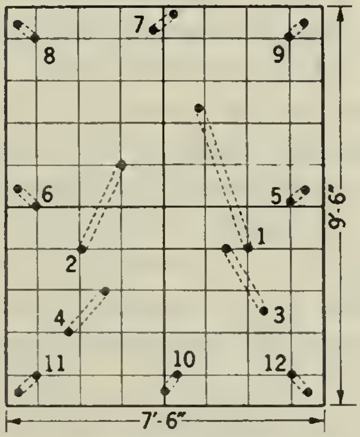
Nature of round, type A					
Hole No.	Depth, feet	Angle Vert.°	Angle Hor.°	Powder, sticks	Firing order
1	6.00	25	10	6	1
2	5.25	5	5	5	2
3	5.25	5	5	5	3
4	5.25	5	5	5	4
5	5.25	5	5	5	5
6	5.25	5	5	5	6
7	5.25	5	5	5	7
8	5.25	5	5	5	8
9	5.25	5	5	5	9
Total	48.00			46	

Nature of round, type B					
Hole No.	Depth, feet	Angle Vert.°	Angle Hor.°	Powder, sticks	Firing order
1	6.5	30	10	6	1
2	6.0	15	10	6	2
3	5.5	5	5	5	3
4	5.5	5	5	5	4
5	5.5	5	5	5	5
6	5.5	5	5	5	6
7	5.5	5	5	5	7
8	5.5	5	5	5	8
9	5.5	5	5	5	9
10	5.5	5	5	5	10
Total	56.5			52	

NOTE: Each square equals 1 square foot



TYPE C, 11 HOLES

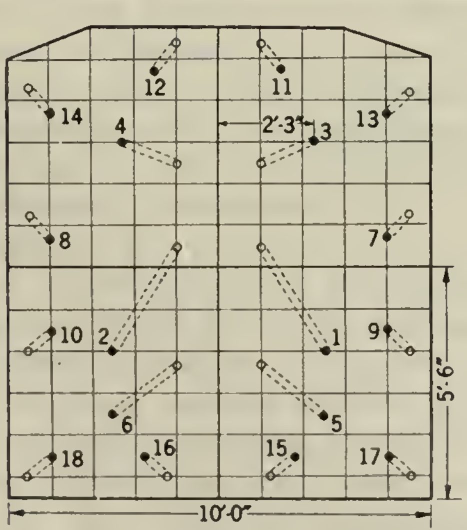


TYPE D, 12 HOLES

Nature of round, type C					
Hole No.	Depth, feet	Angle Vert.°	Angle Hor.°	Powder, sticks	Firing order
1	6' 6"	30	10	6	1
2	6' 0"	15	10	6	2
3	5' 9"	10	10	6	3
4	5' 6"	5	5	5	4
5	5' 6"	5	5	5	5
6	5' 6"	5	5	5	6
7	5' 6"	5	5	5	7
8	5' 6"	5	5	5	8
9	5' 6"	5	5	5	9
10	5' 6"	5	5	5	10
11	5' 6"	5	5	5	11
Total	62'3"			58	

Nature of round, type D					
Hole No.	Depth, feet	Angle Vert.°	Angle Hor.°	Powder, sticks	Firing order
1	6' 6"	30	10	6	1
2	6' 0"	20	10	6	2
3	6' 0"	15	10	6	3
4	5' 9"	10	10	6	4
5	5' 6"	5	5	5	5
6	5' 6"	5	5	5	6
7	5' 6"	5	5	5	7
8	5' 6"	5	5	5	8
9	5' 6"	5	5	5	9
10	5' 6"	5	5	5	10
11	5' 6"	5	5	5	11
12	5' 6"	5	5	5	12
Total	68'3"			64	

Figure 2.—Drift rounds



Nature of round					
Hole No.	Depth, feet	Vertical angle	Horizontal angle	Sticks of powder	Order of firing
1	6' 3"+	25°	15°	6	1
2	6' 3"+	25°	15°	6	2
3	5' 9"	5°	15°	6	3
4	5' 9"	5°	15°	6	4
5	6' 0"	10°	15°	6	5
6	6' 0"	10°	15°	6	6
7	5' 6"	5°	5°	5	7
8	5' 6"	5°	5°	5	8
9	5' 6"	5°	5°	5	9
10	5' 6"	5°	5°	5	10
11	5' 6"	5°	5°	5	11
12	5' 6"	5°	5°	5	12
13	5' 6"	5°	5°	5	13
14	5' 6"	5°	5°	5	14
15	5' 6"	5°	5°	5	15
16	5' 6"	5°	5°	5	16
17	5' 6"	5°	5°	5	17
18	5' 6"	5°	5°	5	18
Total	102'0"			96	

Figure 3.—Eighteen-hole round, motor haulage drift

DRAW CONTROL

In mining by caving methods, draw control is one of the most highly important factors in successful operation. At the Braden mine the draw control is in direct charge of the efficiency engineer. Accurate graphs are made showing the draw from each pair of chutes in the mine. For general purposes, these are brought up to date twice a month, but for the closer guidance of the mine operators special large-scale charts, kept in the mine offices, are plotted daily. Chutes that are lagging due chiefly to timber trouble, are more heavily drawn when the timber repairs are completed, the overdrawn chutes being held back in order that the capping may be brought down evenly. Instructions are given to the operating force each day as to the tonnage to be pulled from each chute.

DEVELOPMENT WORK

The Teniente mine has been developed chiefly from adits, shafts, and inclines. Figure 1 shows a longitudinal section. Due to the probable bottom of the orebody being accessible by means of a tunnel, Teniente 5, the ore is dropped from the mining levels down inolined ore passes (raised up through the orebody) to the retram level, Teniente C, where it is motor trammed to other ore passes located in a hard breccia, highly resistant to abrasion, outside of the ore limits, and thence dropped down these latter ore passes to the main haulage level, Teniente 5. Shaft A is used for handling men and supplies from Teniente 5 to Teniente C. Inclines 2 and 4 likewise serve the levels above Teniente C.

The drifts and crosscuts from which the stoping, pillar caving, and undercutting are done are practically all driven in andesite. Standard rounds, determined experimentally, are universally used. The number of holes in these rounds (fig. 2) varies from 9 to 12, depending on the nature of the ground.

In the larger-sized motor-haulage drifts and crosscuts an 18-hole round is used (fig. 3).

All development work is contracted for on a meterage basis, uniform prices having been adopted for each type of drive.

Drilling.- One-man, mounted jackhamers are used in driving practically all horizontal headings, the drilling being done dry. Stopers (hand-rotating type) are employed exclusively in raising.

Drill Steel.- The jackhamers are all operated with $\frac{7}{8}$-inch hexagon hollow steel, 1-$\frac{1}{4}$-inch cruciform steel being used with the stopers. The gage of the starters is 2 inches, with a reduction of $\frac{1}{8}$ inch for each succeeding bit. The difference in length between each change is about 12 inches.

Drill Bits.- Double-taper rose bits with angles of 14 and 5° are used. All bits are carefully inspected daily, after sharpening, for gage, sharpness, and form of bit. All steel sharpening is done on contract in underground blacksmith shops equipped with oil forges and pneumatic sharpeners.

Air Pressure.- The air pressure at the compressors is 100 pounds per square inch. At the drills it varies from 80 to 85 pounds.

Blasting.- Primers with fuse and detonators are made up by inserting the cap into the end of the primer cartridge. The primer is always the second stick from the bottom of the hole. All blasting is done with 25 per cent N. G., Ammon-Gelignite. For the standard rounds 1$\frac{1}{4}$ by 8 inch cartridges are used. Smaller sticks, $\frac{7}{8}$ inch by 8 inches, cut in half, are used for blockholing, bulldozing chutes, and unchoking ore passes. All blasts are detonated with No. 6 lead azid detonators, which have the same strength as standard No. 8 fulminates. These detonators were adopted as standard after a long series of comparative experiments had been carried on. Hot wire lighters are in general use in spitting the fuse. Moist clay cartridges having the same dimensions as the dynamite sticks are used for stemming. These cartridges are made in the mine in a pneumatic machine which forces the clay into waxed paper tubes, thus forming the cartridges.

DRIFTS AND CROSSCUTS

There are three general types of drifts and crosscuts. (1) For hand tramming, 7 feet 6 inches by 9 feet 6 inches in cross section. (2) For motor haulage, 10 feet by 10 feet in cross section. And (3) for access to control gates and the caving sections for the handling of supplies, 7 feet 6 inches by 9 feet 6 inches in cross section, the same size as for (1) hand tramming; these latter workings being so located when in ore on the lower levels that they can later be utilized as extraction drifts when the mining operations progressively reach the lower horizons.

Hand Tramming Drifts and Crosscuts.- Hand tramming drifts and crosscuts are broken full size to eliminate as far as possible the necessity of widening the drift in order to install the chute sets (fig. 4).

The depth of the round broken averages 1.47 meters (4.82 feet).

The mucking is done by hand into 1-ton cars and on the night shift, the drilling and blasting being done on the day shift. Experiments are being carried on with different types of scrapers and slushers.

Motor-Haulage Drifts and Crosscuts.- Motor-haulage drifts and crosscuts are always timbered wherever the ground shows any signs of sloughing, either from the back or sides. As has been noted, a standard 18-hole round is used, shown in Figure 3, the drilling being done with one-man, mounted jackhamers using $\frac{7}{8}$-inch hexagon hollow steel.

The mucking here is also done by hand into 5-ton cars. Some experimentation is being carried on with mechanical loaders.

Drifts For Access to Control Gates, Handling Supplies.- In so far as their driving is concerned, drifts for access to control gates can be classified with those for hand tramming.

RAISES

Ore Passes.- At Braden it has been found that the ore transfer raises or ore passes, as they are locally known, operate best when driven on an angle of about 60° with the vertical.

These ore passes are classified as: (1) Main ore passes, those located in the hanging-wall breccia and running from Teniente 5 to Tiente C; and (2) retram ore passes, those running from the retram level to the producing level, generally through the ore-body. The accompanying sketches (fig. 5) show each type.

Main Ore Passes.- Main ore passes and their auxiliaries are always raised in the hanging wall, in a tough breccia, which, as has been noted before, is highly resistant to the abrasion of the falling ore. These raises are run 2 by 2 meters (6.56 by 6.56 feet) in cross section. The round consists of 16 holes loaded with approximately 94 sticks of 25 per cent N.G., Ammon-Gelignite, giving a break of 1.20 to 1.50 meters (4.25 to 4.92 feet). Stoper machines are used, two miners working together and sharing in the contract, which is let on a meterage basis.

The general method of raising one of these ore passes is as follows:

When the ore pass has attained a slant height of 5 meters (16.42 feet) above the level, the miners drive two holes in the floor, normal to the floor, 1.5 meters (4.92 feet) down from the face, into which they insert two pieces of 1-inch round iron, leaving 2 feet projecting out in the ore pass. Against these supports is placed a 2 by 12 inch by 5 foot board which serves as a platform for the miners and their machines while drilling the round. Attached to one of the above pieces of projecting iron is a $1\frac{1}{2}$-inch diameter hemp rope, with the help of which the miners climb up the floor of the ore pass to the working face. As the raise advances, the platform with its iron supports is moved up, being kept at all times

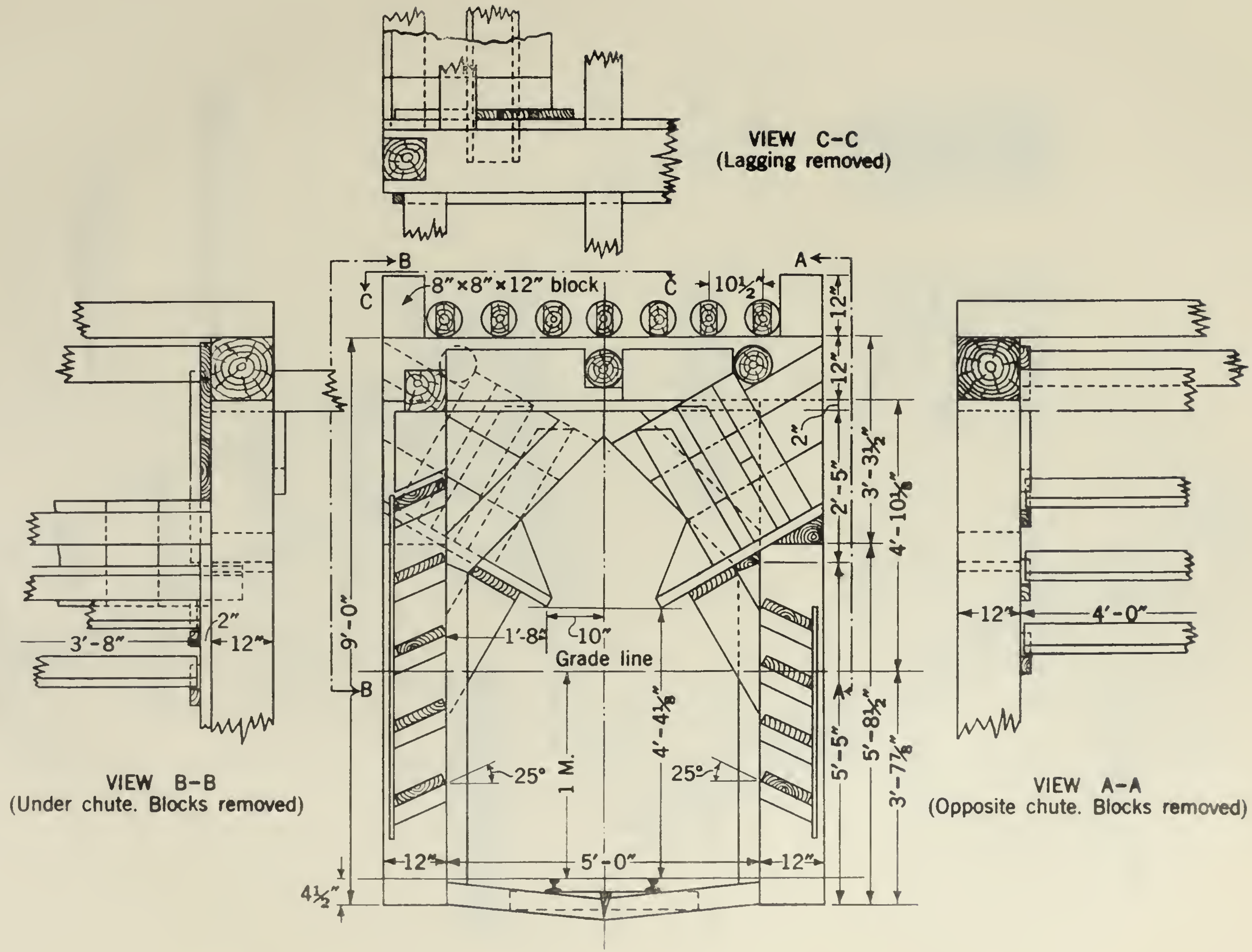


Figure 4.—Standard chute set

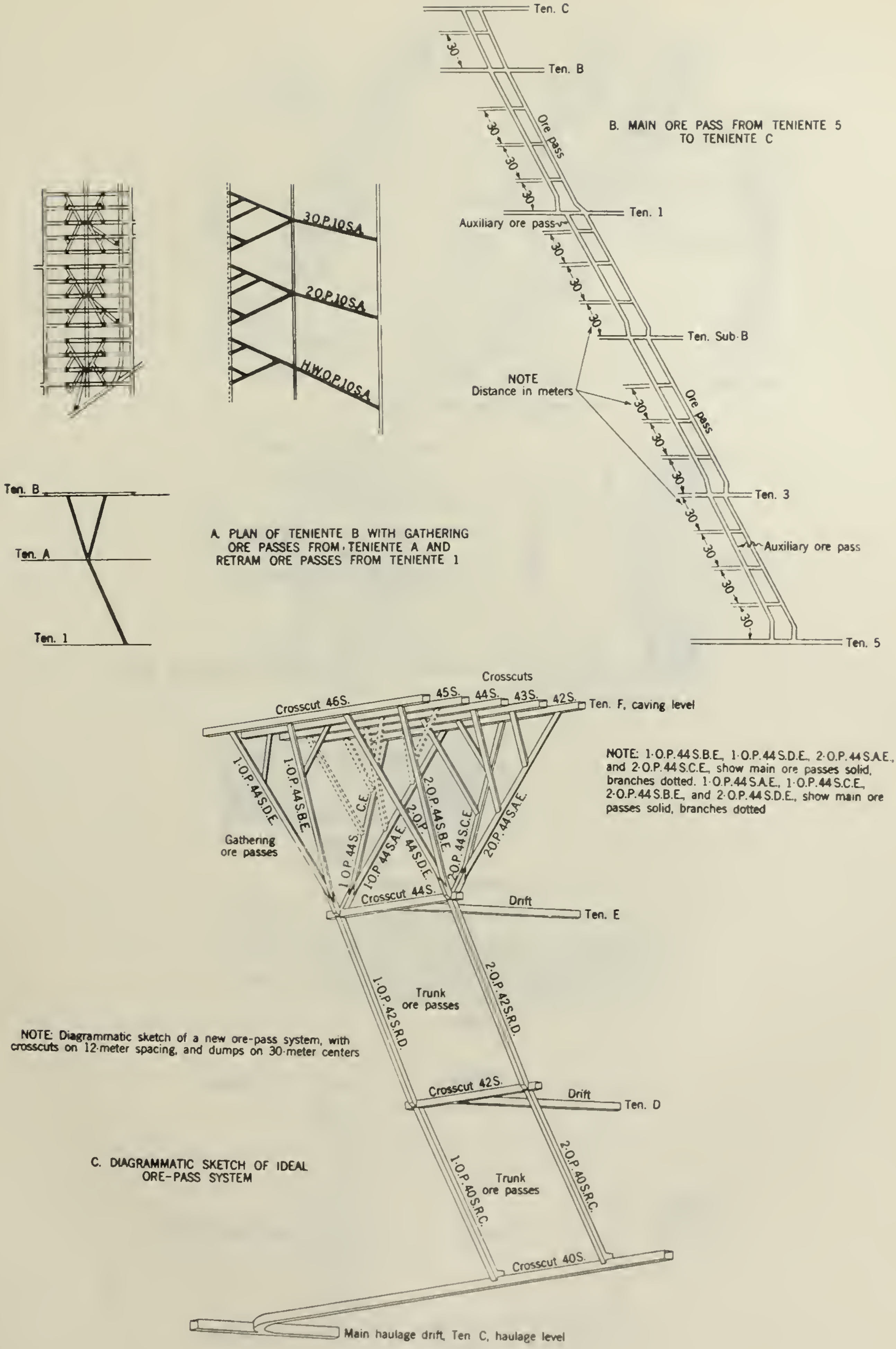


Figure 5.—Types of ore passes

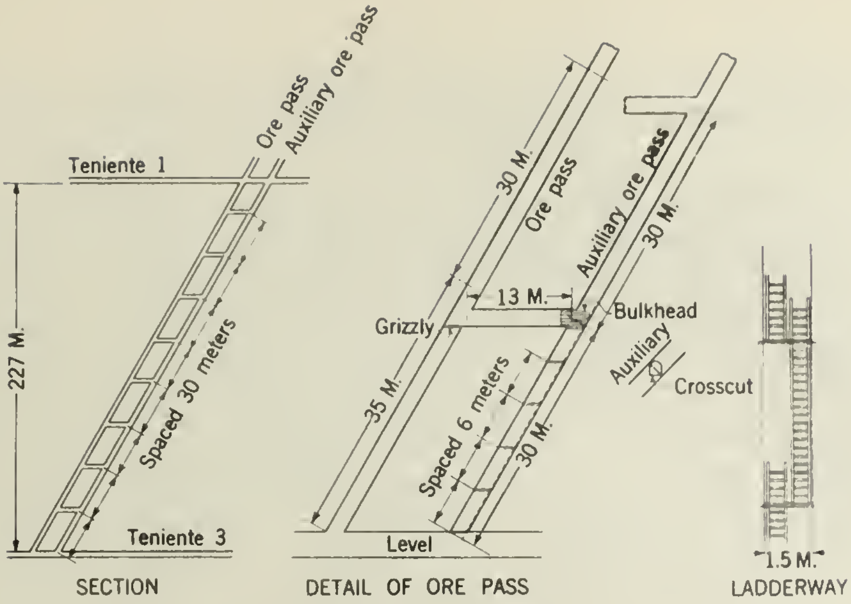


Figure 6.—Method of driving ore pass

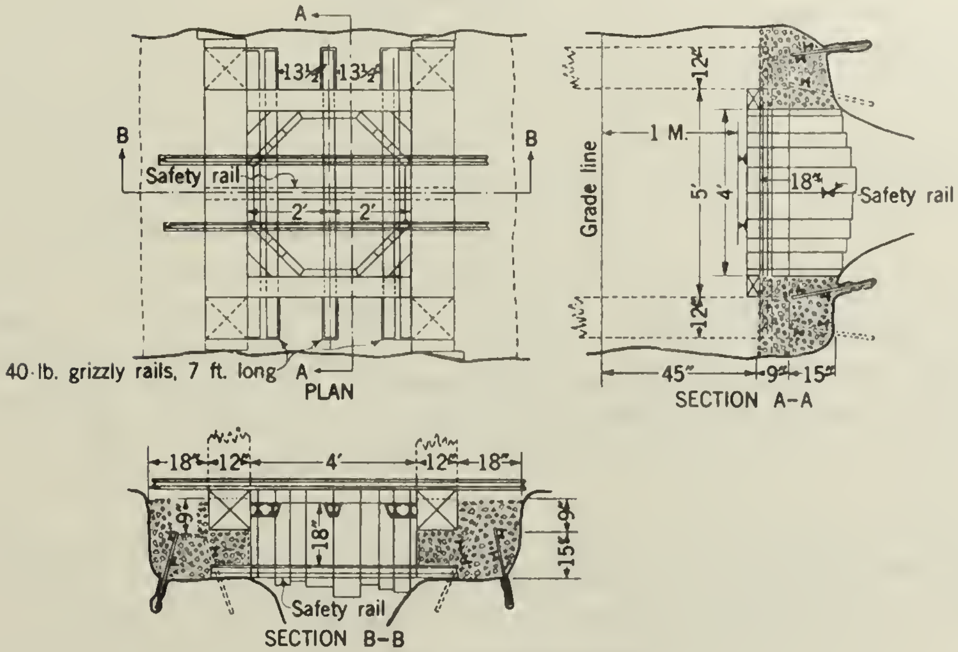


Figure 7.—Standard grizzly

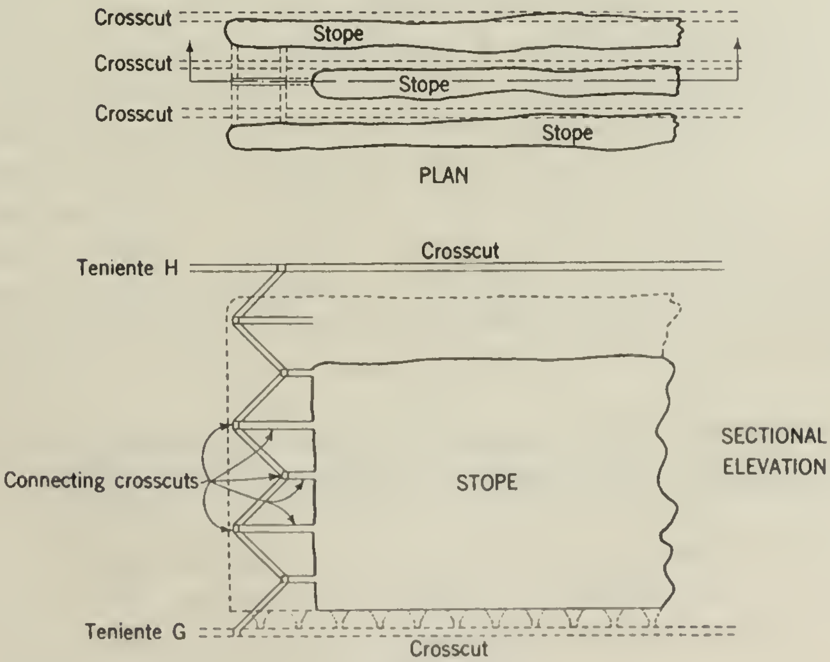


Figure 8.—Turret raise

within 1.5 meters (4.92 feet) of the face. The compressed air to supply the machines is carried up the raise in a 2-inch air line, which is progressively extended to within 20 feet of the face as the raise advances.

When the ore pass and auxiliary have both been raised for a distance of 35 meters (114.75 feet) they are temporarily stopped and a connecting crosscut is driven (fig. 6). When this connection is made, a grizzly is set in the ore pass, level with the floor of the crosscut. In the auxiliary a bulkhead is placed 2 feet above the floor of the connecting crosscut and extended into it, enabling the miners to get from the auxiliary, underneath the bulkhead, into the crosscut. This bulkhead further serves as a floor on which the broken material from the blasts in the auxiliary is caught. This muck is trammed through the connecting crosscut and dumped into the ore-pass grizzly. A ladder way is placed in the auxiliary from the level to the bottom of the bulkhead, affording a convenient entrance for men and supplies. These connecting crosscuts are driven at intervals of 30 meters (98.4 feet) and the same procedure is continued throughout the raising of the ore pass and the auxiliary. The ultimate use of the auxiliary in addition to the safety feature during the driving as outlined above, is to provide a secure means of access for inspection of the ore pass and to enable chute blasters to place charges in the ore pass when it becomes choked due to an accumulation of fines or too high a moisture content in the ore. As soon as an ore pass has been driven up a few meters, a "Cousin Jack" chute is put in on the haulage level to eliminate any shoveling. The ore pass is connected from the level below to the one above, and a grizzly is placed before continuing with the raise, thus eliminating any handling of the muck on the intermediate levels.

Retram Ore Passes.- Retram ore passes are driven in the same manner as those on the lower levels and with the same cross section. However, due to the shorter lifts the upper level raises are put up without the accompanying auxiliaries and the small connecting crosscuts. As the retram ore passes are generally located in andesite, a 5-foot round can usually be broken with nine holes, each loaded with a charge of 5 sticks of 25 per cent N.G., Ammon-Gelignite.

Winzes to connect with these ore passes are sunk from the level above to a depth of 2.5 meters (8.2 feet). The winzes are driven as small as possible and enlarged, after holing through, to such size as will permit the permanent grizzly with its concrete reinforcement and safety rail to be securely and accurately placed (fig. 7).

The object of driving the winze of extremely small size is to avoid a large overbreak in the drift or crosscut, which would necessitate a very large grizzly sill and introduce the possibility of excessive maintenance cost due to the grizzly sill taking weight. Under the present method of sinking very small winzes and later enlarging them, after the connection has been made the grizzly sills are set on solid ground, and only in exceptional cases are failures experienced.

Exploration Raises.- Exploration raises are put up on a 45° slope with a cross section of 1.50 by 1.50 meters (4.92 by 4.92 feet). Wherever practicable they are planned to be used later as observation raises for noting any caving action of the block undercutting; and also when necessary for ventilation, they are broken through to the surface or connected with the level above.

Pilot Raises.- Pilot raises are used in making shaft connections. The shafts are located in a hard breccia, and the method of raising, in a large-sized shaft, is as follows:

The pilot raise is zigzagged up on a 50° slope, being kept within the shaft section. After breaking through to the top level the raise is enlarged to the full shaft size from the top down, the shaft timber being carried down with the enlargement, and the spoil dropping down through the pilot raise to the level below.

SHAFTS

Shafts 1 and 2 were sunk and raised to provide for handling men and supplies, between Teniente 5 and Teniente C, during the development period of the Teniente mine.

Shaft A was completed in 1920 to take care of the increased traffic for mine production. This shaft has two hoisting compartments 9 by 7 feet 1 inch in size each equipped with a 50-man, double-deck cage and a manway of 9 by 6 feet, up which are carried air, water, and power lines and a ladder way. The shaft is timbered with 10 by 10 inch Oregon pine sets, spaced on $6\frac{1}{2}$-foot centers. As a protection against fire this shaft is equipped with an automatic sprinkler system. The hoisting equipment consists of a double drum hoist, operating in balance, driven by a 160-hp. A.C. motor. The drums are 7 feet in diameter by 4 feet 6 inches long and are grooved for $1\frac{1}{8}$-inch cable. This equipment is provided with both overwind and overspeed safety devices.

MINING METHODS

In the early days the higher-grade portions of the outcrops were first worked intermittently, all ore being hand sorted and sacked. Later, the high grade having become depleted, a tunnel was driven in Fortuna 2, and a raise put up to the surface. This showed ore running about 4 per cent copper, but as the ore was well disseminated, it was impossible to handle it at a profit without concentration. Mr. Braden, becoming interested at about this time, put up a 250-ton mill and ore was obtained first by glory-hole methods and then, as this gradually became impracticable, by shrinkage stoping.

The original Fortuna stopes were planned to run normal to the long axis of the orebody with a width of 7 meters (22.95 feet) having a 10-meter (32.8 feet) pillar between them. In the center of these pillars starting alternately from the footwall and hanging-wall boundaries the stope entrance chimneys were raised on a 45° slope, zigzagging, in the plane of the pillar, from the footwall toward the hanging wall or vice versa. These chimneys gave access for men, supplies, and the air line to the shrinkage stope on both sides by means of small crosscuts spaced 7 meters (22.95 feet) vertically. The height of these shrinkage stopes ran up to approximately 100 meters (328 feet). It was found, however, in subsequent pillar caving operations, that it was difficult to break such thick pillars, due to their tendency to arch longitudinally so as to necessitate raising small stopes in the footwall and hanging-wall extremities of the pillar. Because of this unfavorable condition, when stoping was started in the Teniente mine the thickness of the pillars was decreased to 5 meters (16.4 feet), leaving the width of the stope at 7 meters (22.95 feet). This required a change in the stope entrance chimney. A "turret raise" was then substituted. This raise was driven on a 45° slope with 8-meter (26.23 feet) legs, one raise being located at each end of a set of three stopes and serving them by small connecting crosscuts (fig. 8).

The successful caving of these pillars still presented difficulties, and a further reduction in their width was made to 3 meters (9.84 feet), reducing the stope width to 5 meters (16.4 feet).

Cribbed manways were then adopted for stope entrances, these being raised in solid rock at each end of the stope and abutting the slope. These manways are equipped with ladders and are used as an entrance for men, supplies, and a 2-inch air line. At the foot of the manway a tugger hoist is located, by means of which steel and other supplies are hoisted to the top of the stope (fig. 9).

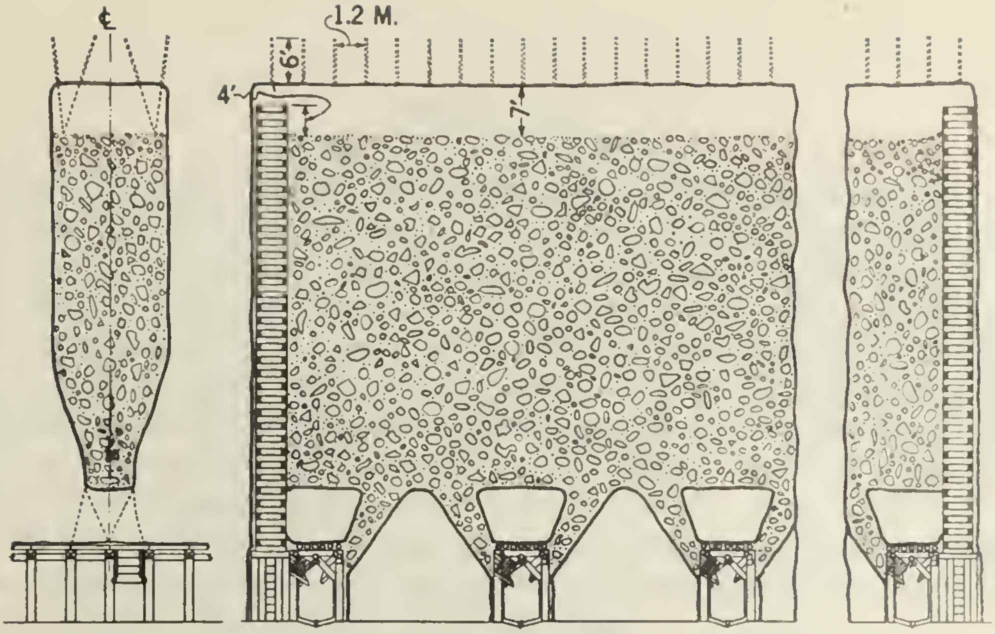


Figure 9.—Spacing of holes in stope

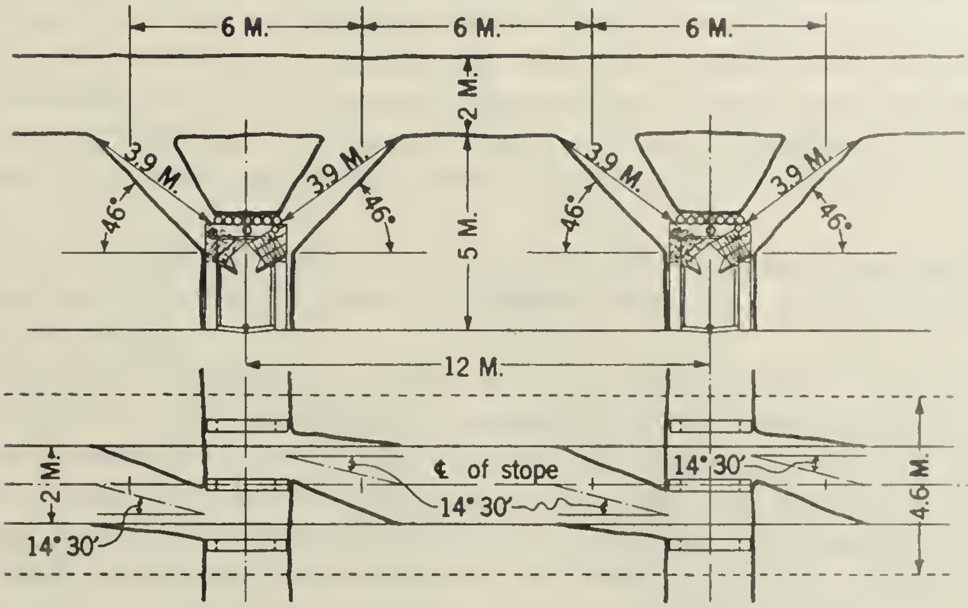


Figure 10.—Stope chute raises. Stope raised on new system normal to drift

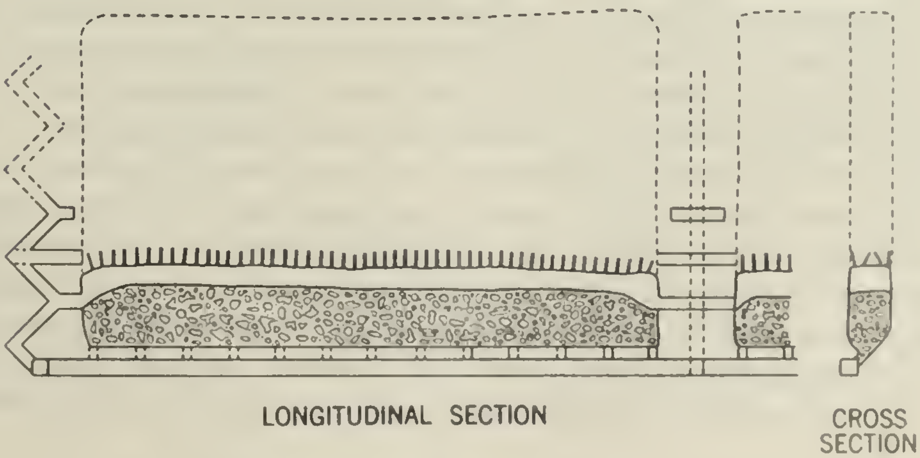


Figure 11.—Stope raised on old system parallel to crosscut

STOPING

The drifts under the stope having been driven, the first step is to put up the stope chute raises from these drifts (see fig. 10). These are located on 6.1 meters (20 feet) centers and are driven up 3.90 meters (12.8 feet) on a 46° slope at an angle of 75°30' from the center line of the drift to the elevation of the stope sublevel. After driving about 1 meter (3.28 feet) of the stope chute raise (one round), the timber for the chute is stood and the chute box installed. The tops of the stope chute raises are then connected by a drift, 7 feet high and 6 feet wide, from the hanging-wall to the footwall limits of the stope. The chutes are then belled out to insure good drawing, and the stope is started by widening out the drift and raising from it.

Approximately 40 per cent of the tonnage of the ore broken out of the solid is drawn for production, the remaining 60 per cent being left in the stope to support the pillars and allow the miners to drill the back for the next round.

The stope crew drill and blast 50 meters (164 feet) along the stope during each shift. The number of miners required is roughly calculated at 10 per stope, thus giving each pair about 10 meters (32.8 feet) of back to drill. The holes are spaced along the length of the stope from 1 to 1.25 meters (3.28 to 4.1 feet) apart; they are drilled in sets of four consisting of two cut holes and two side holes, the cut holes being given an inclination of about 20° from the vertical and the side holes being sloped slightly toward the walls. The average depth of hole is 6 feet and the average number drilled per machine shift varies from 15 to 20. An average of 70 tons per machine shift is broken, of which 28 tons can be drawn as shrinkage.

All the men in the stope, including the stope boss, stopemen, mechanics, and tool nippers, work on contract, the monthly basis of payment being the amount of broken tonnage as calculated by the engineers.

With the changes in the sizes of the stopes and pillars the system of crosscuts driven from hanging wall to footwall was changed to a system of drifts running north and south or perpendicular to the crosscuts. The orientation of the stopes remained as before, thus bringing them normal to the drifts. The basis of this change was the theory that the compressive stresses set up by the caving of the pillars, during the pillar caving operation, crushed the pillar stubs, in stopes raised parallel with the crosscuts, thereby throwing the entire overhead weight on the crosscut timbers. With the stopes normal to the drifts it was reasoned that the lugs between the drifts would not break up as readily and would carry more of the weight of the broken mass, thus relieving the pressure on the drift sets and decreasing the timber costs. Actual practice has borne out this assumption, by a reduction in the cost of timber maintenance. A further advantage of having the stopes run normal to the drifts is the greater amount of ground available at all times for extraction. With the old system (fig. 11) it was necessary to cave from 60 to 90 meters (197 feet to 295 feet) of the crosscut before draw operations could be started. In heavy ground often as much as half of this available drawing area was down due to timber failures in the crosscut. This greatly reduced production from the section under draw. With drifts, and stopes normal to the azimuth of the drifts, it is seldom necessary to have over 30 chutes of 45 meters (147.6 feet) undercut at one time. Caving from hanging wall to footwall, the various drifts give more points of access for drawing. If one or more drifts have timber trouble, seldom, if ever, is the work of more than two loaders interfered with, whereas under the old system from six to ten loaders might be unable to draw in the section affected.

PILLAR CAVING

The work of pillar caving, consisting of breaking up the solid pillar between the stopes, is commenced by putting up the chute raises 3.50 meters (11.48 feet) into the pillar. The tops of these raises are then connected by a 2 by 2 meter (6.56 by 6.56 foot) drive for the entire length of the pillar. Over a 6-meter (19.67-foot) length the top and sides of this drive are enlarged, and holes are then drilled to within breaking distance of the adjoining stopes. These holes are blasted, with the resultant breaking up of the base of the pillar. This procedure is continued for the full length of the stopes until the entire pillar base is weakened. The subsequent drawing of the chutes, underneath, furthers the complete disintegration of the pillar. This process is graphically shown in the accompanying sketches. (Fig. 12.)

BLOCK UNDERCUTTING

The block undercutting method of mining is the gradual development of a system evolved at Braden from the experience gained in the caving of pillars between shrinkage stopes. The method can be defined as the process of caving virgin ground adjacent to country that has previously been weakened or caved by a minimum amount of stoping and pillar caving.

In shrinkage stoping, as has been noted before, it is necessary to raise stope chutes and connect their tops with a small crosscut. With the present caving practice this raising and driving is eliminated, resulting in a corresponding saving in the preparation work. This excessive preparation work and the high cost of breaking ore in stoping were the main factors in bringing about the change in caving methods, the new system being obviously more economical. The cost of ore broken in shrinkage stoping has averaged 43.37 cents per ton while block undercutting costs have averaged 20.5 cents per ton, a direct saving of 22.87 cents. With this marked reduction in cost by block undercutting as compared with stoping, the present practice is toward raising a minimum number of shrinkage stopes, mainly for boundary cut-offs and such preliminary stope weakening as is necessary on a new level. Of the present delivered tonnage, over 85 per cent is derived from block undercutting.

The undercutting is commenced by taking a section of a drift 6 meters (19.68 feet) long and drilling a 6-foot round above the top lagging and through the drift chutes, as shown in Figure 13 at A. The ore broken in this first round is drawn out through the drift chutes, leaving a small open stope approximately 7 feet high above the drift timbers. Access to this stope, after the first blast, is through the end chute next to the virgin country in the section of drift blasted. Preliminary to this initial blast, an entrance chimney has been completed from the adjacent drift to the center of the pillar, as shown in the sketches. This entrance is driven with a 1.50 by 1.50 meter (4.92 by 4.92 feet) section on a 48° slope with an angle of 8° 30' from the perpendicular to the drift to meet a similar chimney driven from the last chute in the undercut stope, as shown. Before making the second widening blast in the undercut stope, the entrance chimney is connected, thus providing a safe entrance for men for subsequent shots in the undercut stope. A second and third widening blast are made, as shown in Figure 13, B and C. Upon making the third blast, the ground usually shows signs of weakening, and it is necessary to stull up the back in order to assure safety for the miners in drilling and loading the final blast which breaks to the adjacent caved country, as shown in Figure 13 D. This operation is carried on in progressive slices in successive drifts until sufficient country is weakened to produce continuous caving action. After an undercut section has been completed in a drift, 300 tons of ore per chute are drawn to allow sufficient room for further caving action. These chutes are then sealed and remain closed until the next progressive slice has been undercut. The reason for

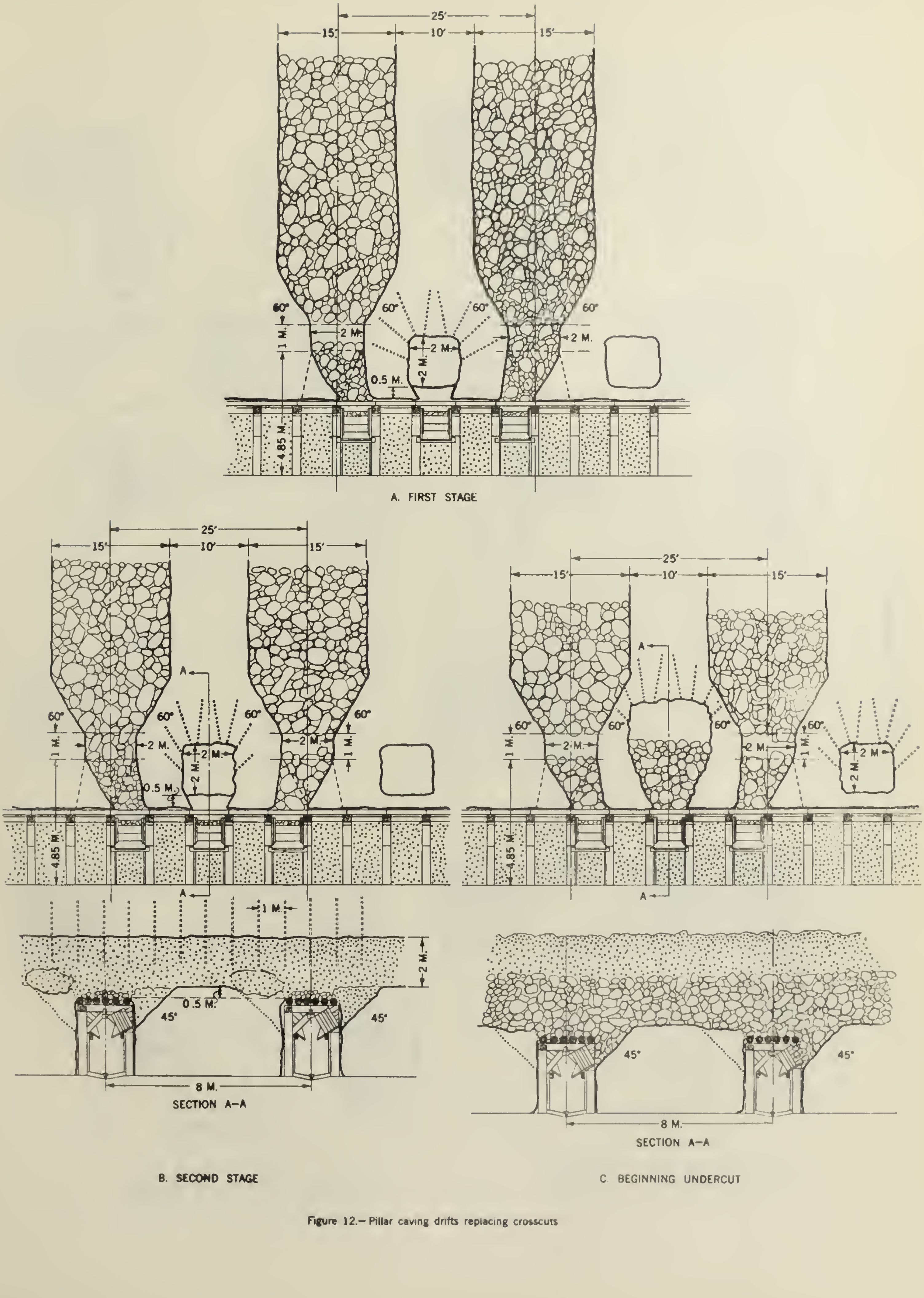


Figure 12.– Pillar caving drifts replacing crosscuts

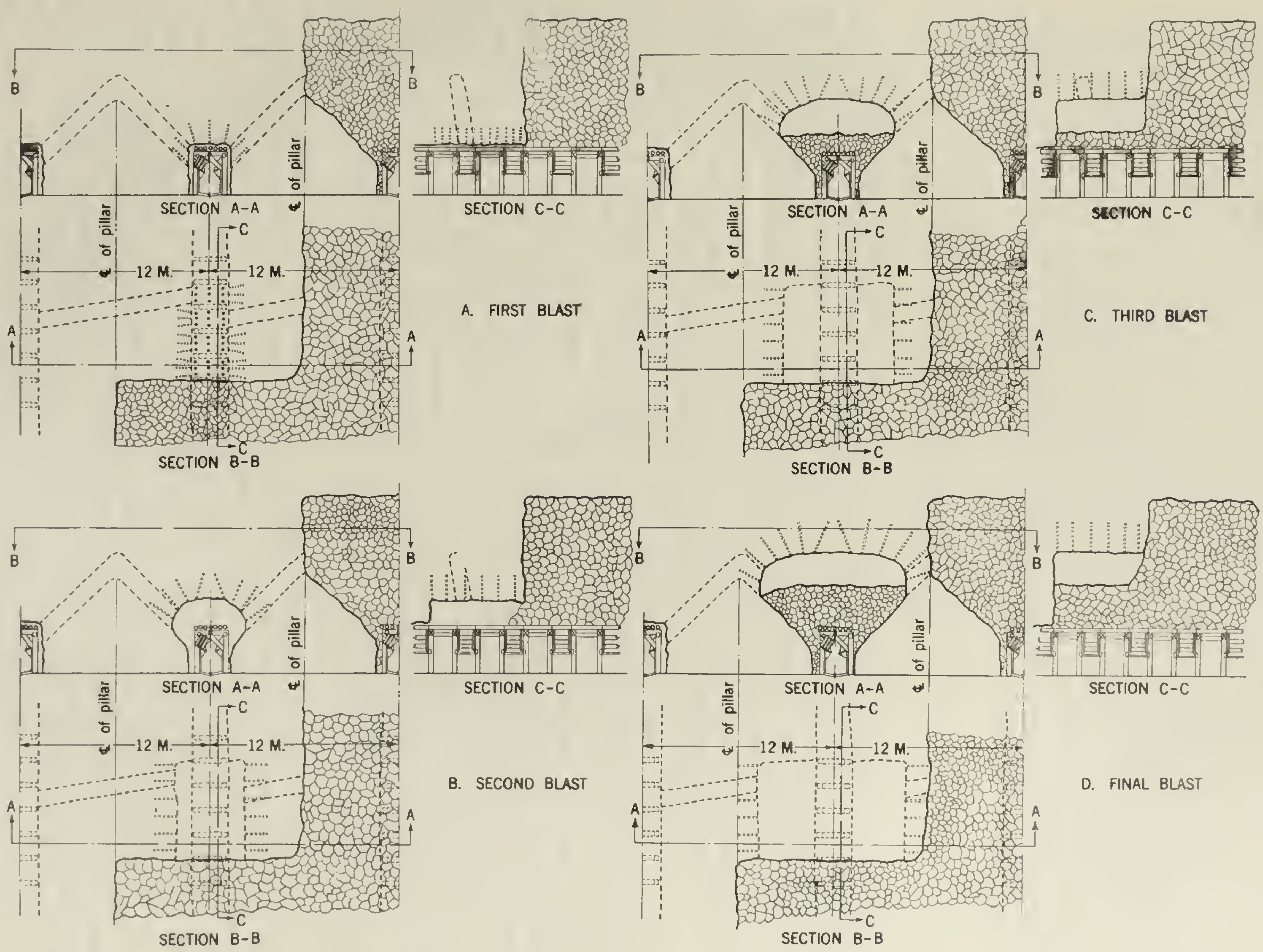


Figure 13.—Undercutting practice

this procedure is to keep the miners from working against an open stope. That portion of the entrance chimney, leading into the last undercut stope, is used as an observation entrance to note the caving action in the preceding slice. Where a pillar is unavoidably left between this undercut slice and the preceding slice, and where the back is firm enough to allow stulling, a passageway is opened up to allow redrilling the unbroken pillar.

An undercut crew consists of four miners who are paid on contract on a lineal meterage basis of drift undercut. Stoping machines are used exclusively for all drilling.

Formerly it was considered at Braden that 50-meter (164 foot) lifts were the maximum that could be caved successfully. Due to a study made on the action of caving ore, however, 100-meter (328 foot) lifts were attempted and successfully caved, with no loss of ore and with no excessive waste dilution. All present development is laid out on the basis of 100-meter lifts.

UNDERGROUND HAULAGE

Hand Tramming.- All hand tramming is done by means of 1-ton cars. The trammers are placed so as to give each one a minimum of 10 chutes to pull in order that a daily tonnage, specified by the draw engineer, may be drawn from each chute. Dumps are spaced on 30-meter (98.4-foot) centers in the drifts, thus giving each trammer a maximum tram of 15 meters (49.2 feet). An average of 55 tons per trammer is produced per 8-hour shift. Chute blasters aid drawing operations by bulldozing the chutes and generally helping the trammers. Usually one chute blaster is employed to every 3.5 to 4 trammer, the ratio depending on the kind of ore being drawn. Twenty-one-pound baby jackhamers are used in bulldozing.

Since the payment for hand tramming in caving sections is made on a contract basis it was necessary to have an accurate count of cars trammed by each man. Formerly this record was kept by car-checkers. This system proved unsatisfactory and an automatic car counter, operated by an arrangement of springs and cams, was designed and installed on each car used in the caving sections. As the cars are of 1-ton capacity, it is necessary for the trammers to load this weight into the car and actually dump it before the automatic device will register. The markers on each car are read at the beginning and end of each shift. The number of cars paid for is based on the difference in the readings of the counters. The success of this automatic counter is borne out by the fact that the produced tonnage over the past two years has been equal to the weighed tonnage delivered to the mill.

Electric Motor Tramming. (Teniente C).- The ore is retrammed on Teniente C level, where it is collected from the gathering ore passes which serve the producing levels, and is delivered to the main hanging-wall ore passes. The latter run down to the main electric haulage way on No. 5 level, whence the ore is delivered to the coarse-ore bins (fig. 1).

The retram haulage equipment consists of 10-ton locomotives, driven by D. C. motors, having a speed of 8 miles per hour and a capacity load of fifteen 5-ton ore cars per train. The cars are of the gable-bottom type with an overall length of 11 feet 7½ inches, width of 5 feet 2 inches and a capacity of 95 cubic feet. They are provided with either hand or automatic air dumpers. Each electric locomotive is equipped with an air compressor, type CP 28-A, compressing to 90 pounds, having a capacity of 25 cubic feet of free air per minute and discharging into an air receiver having a capacity of 7.5 cubic feet. From this receiver air connections are made with the train through standard railroad air-line equipment. On one end of each car is an air cylinder, controlled by a 3-way valve, the piston of which actuates a ratchet arrangement on the end of the piston arm engaging with a half gear. This half gear is connected to the door lever arms and controls them. By standing at the side of the train and operating the 3-way valve a man can open and close the side of the cars over the main ore pass dumps, thus eliminating the dangerous practice of climbing in between

the cars and manually operating the dumping device. The time consumed in dumping the trains is also lessened, having now been decreased to one minute. Roller bearings are standard equipment for this section. The track has a gage of 30 inches, 40-pound rail being used. All switches throughout the level are governed by semiautomatic signals. The standard loading gate used on the level allows a 15-car train to be loaded in four minutes. The dumps on this level are 17 feet long and 14 feet wide. The grizzlies are made with 60-pound rail spaced to give a 14-inch opening. Train movement throughout the level is controlled by a train dispatcher. A special telephone system, installed exclusively for this service, has connections at each crossing, ore pass, and dump. By means of this system, train orders are communicated to the train crews.

Operations are conducted on two 8-hour shifts with four 15-car trains, allowing a 4-hour interval between shifts in which to take care of such items as track maintenance and cleaning up. Payment of motor crews is by contract based on monthly tonnage hauled. The handling of the ore through the ore passes above Teniente C is controlled by gate tenders working on the intermediate levels and under the supervision of the retram foreman. Figure 14 shows the type of control gate used on the intermediate levels.

Electric Motor Tramming. (Teniente 5).- On Teniente 5 level the ore is transported from the main ore passes, coming down from Teniente C, to the mill coarse-ore bins located on the surface. The gage of the track in this haulage way is 30 inches but 60-pound rails are used to take care of the heavier traffic. The grade is 0.5 per cent in favor of the loaded ore trains. Twenty-five-ton electric locomotives, operating on 550-volt direct current, having a maximum speed of 7 miles per hour with a drawbar pull of 9,000 pounds at this speed, and equipped with air and hand brake control, are used exclusively in this section of the mine. The maximum ore haul is 3,886 yards. Trains make a complete round trip, including weighing and dumping, from the main ore passes to the mill bins in 30 minutes The ore cars used are of the gable-bottom type, having a capacity of 23.8 tons. The trains are equipped with automatic air control. They consist of 12 cars and operate singly on the return trip from the mill bins to the ore passes and in tandum on the outgoing run.

The train crew consists of a motorman and two brakeman, who assist in loading at the ore passes and in the dumping operation at the bins. All switches are governed by electric semaphore signals and train movements are controlled through a central dispatching office, where each train crew reports to receive train orders.

Supply and passenger trains are also operated at regular intervals over this road. Steel passenger cars, of the Broadway type, are used for passenger service.

HOISTING

Hoisting is confined to the handling of men and supplies, all vertical ore handling being done by gravity through the ore passes.

WAGE, CONTRACT, AND BONUS SYSTEMS EMPLOYED

In 1928 seventy per cent of all underground mine labor was paid on a contract basis.

A standard set of contract prices is in effect throughout the mine, covering all such work as drifting, raising, winzing, stoping, undercutting, tramming, steel sharpening, and timbering, both for standing new sets and for repair work.

All men working on contract are guaranteed day's pay earned at the base rate of the occupation involved.

A bonus is paid on rush work, such as driving drifts or tunnels, where speed is an important factor. This is arranged on a meterage basis, an additional payment being made

for every meter driven over and above the predetermined monthly average for the particular type of ground in which the work is being done.

VENTILATION

No mechanical ventilation has ever been found necessary. Circulation is always good due to the system of ore passes, with their connecting auxiliaries, various shafts and inclines, and adits and raises broken through to the surface.

The air is always cool and the quantity good.

DRAINAGE

All drainage is by gravity to the various portals of the mine, no pumping being required

RESCUE WORK

Classes in mine rescue work are conducted by the safety engineer. Periodic drills, giving instruction as recommended by the United States Bureau of Mines, are given to all bosses.

These drills cover the use of oxygen helmets and all other rescue apparatus.

SAFETY METHODS
(First-Aid Organization and Training)

Accident prevention and the safety of workmen have always been considered as major problems.

First-aid classes are conducted the year round by the safety engineer for the bosses and workmen.

A safety inspector makes daily mine inspections and recommendations with regard to any hazards or dangerous practices observed.

The use of hard boiled hats is compulsory throughout the mine. As a protection against hand wounds, armoured gloves are furnished to all trammers. For work in ore passes and other hazardous locations, the workmen are provided with safety belts. Chute blasters and miners are supplied with canvas bags in which to carry their powder.

A bonus system, based on the severity rate of accidents, has been devised, whereby the bosses have an incentive to enforce all safety rules and measures in vogue in the mine.

SUMMARY OF COSTS

Teniente mine — Year, 1928.
5,174,017 tons delivered
Mining method, block undercutting

	Labor[1]	Air drills and steel[2]	Power	Explosives	Timber	Other supplies	General expense	Total
Development, cents per ton	1.01	0.39	-	0.64	0.33	0.20	2.06	4.63
Mining (undercutting) ...do.	7.77	0.89	-	1.52	4.15	0.63	4.14	19.10
Hand tramming ....do.	3.93	-	-	-	-	0.27	-	4.20
Ore handling[3] ....do.	2.14	-	0.04	-	0.28	0.77	-	3.23
Transportation (E.R.R.)[4] do.	1.00	-	0.10	-	-	0.56	-	1.66
General expense ....do.	-	-	-	-	-	-	15.55	15.55
Total cost.............do.	15.85	1.28	0.14	2.16	4.76	2.43	21.75	48.37

1 Labor includes supervision.
2 Includes power for compressors, maintenance of drilling machines, new drill steel, and cost of drill sharpening.
3 Includes retramming on Teniente C and passing ore from the producing levels to the main haulage level.
4 Includes only electric haulage from main ore pass gates to mill bins.
5 Of total general expense, approximately one-half covers charges applicable to the mine proper, such as sampling, fire protection, safety measures, efficiency, engineering, water, service handling supplies, office and hospital expense, and sundries. The balance consists of outside charges, such as Rancagua and New York office expense, taxes, insurance, welfare, legal expense, etc.

COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Year	Dry tons mined	Tons per man: Direct operation underground	Tons per man: Total pay roll, including surface and shops
1927	5,288,310	13.2	9.2
1928	5,174,017	12.2	8.7

Power, kilowatt-hours per ton shipped

Item	Kilowatt-hours per ton
Retramming (Teniente C)	0.15
Electric railroad	0.34
Air service	1.85
Steel sharpening	0.01
Mine, general (sampling, hoisting, water service, lighting, heating)	0.56
Total	2.91

Explosives
(Ammon-Gelignite 25 per cent N.G.)

	Tons broken per pound of powder	Pounds of powder per ton broken
Primary Blasting:		
Development	0.83	1.205
Undercutting (including stoping)	28.30	0.035
Secondary Blasting:		
Chute blasting on drawing levels	35.7	[1]0.028
Ore-pass blasting	98.6	[2]0.010
	Tons delivered to mill per pound of powder	**Pounds of powder per ton delivered to mill**
Total explosives	8.62	0.116

1 - Bulldozing chutes. 2 - Unchoking ore passes.

Timber

Timber, bd. ft. per ton, 1.08 .... Based on total tonnage delivered to mill and includes timber used in all mine operations.

DETAIL OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Development

	Drifting	Cross cutting	Raising	Total for all develpoment
Specifications:				
Size of excavation	7'6" x 9'6"	7'6" x 9'6"	6'6" x 6'6"	-
Timbered, or not	Yes	Yes	No	-
Physical properties of rock	Medium	Medium	Medium	-
Labor (man-hours per foot):				
Breaking	1.73	1.73	2.10	1.93
Timbering (extraction drifts only)	3.20	3.20	-	3.20
Shoveling	3.27	3.27	-	3.27
Total labor	8.20	8.20	2.10	8.40
Feet per 8-hour shift	4.61	4.61	3.81	4.15
Power and supplies (per foot):				
Explosives (lbs. per foot)	5.50	5.50	4.32	5.18
Timber (board feet, extraction drifts only)	137.8	137.8	-	137.8
Power and supplies, percentage of total cost	-	-	-	25.5

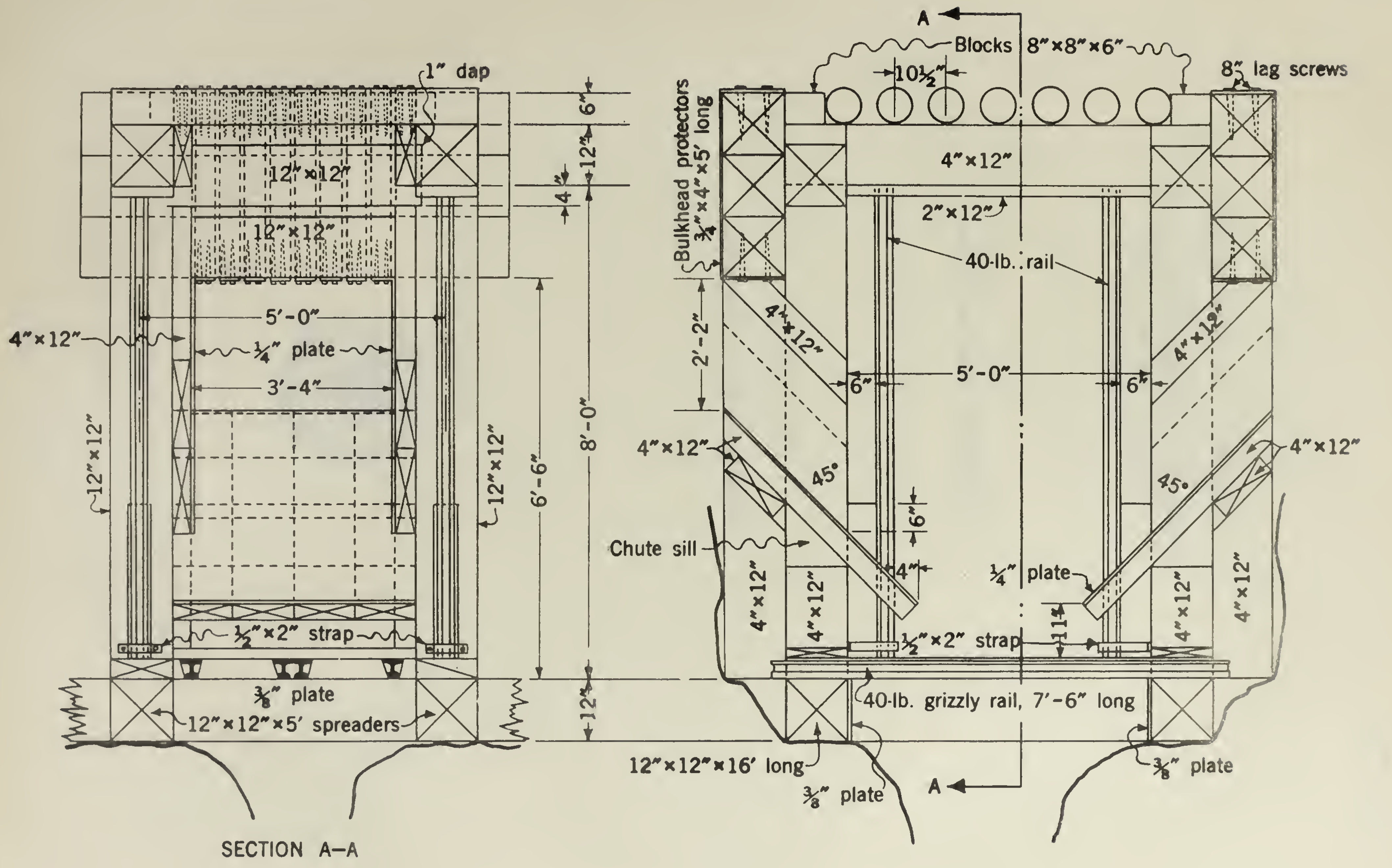


Figure 14. —Standard 2-way ore pass chute control

I. C. 6566 April, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

CHROMIUM

GENERAL INFORMATION

BY

Lewis A. Smith

I.C. 6566
April, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

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CONTENTS

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6566."

2/ Associate mineral economist, common metals division, U. S. Bureau of Mines.

INTRODUCTION

This circular presents a summary of the domestic chromium industry, and discusses the salient features of the world situation, reflecting the literature of the subject. The United States Bureau of Mines has in its files additional detailed information on many subjects briefly discussed in this paper and desires to assist the industries and the public by giving further data in response to individual inquiries. It will welcome comment and criticism and any contributions that would assist in making future editions of this circular more accurate.

Unlike most common metals, chromium is rarely marketed as a pure metal. The chromium industry deals mainly with chromite, the principal ore of chromium, and with ferrochromium, the basic alloy in which chromium enters the metallurgical industry.

PROPERTIES OF CHROMIUM

It has been estimated that about 99.5 per cent of the earth's crust is made up of one-third of the elements. Chromium is one of the large group of relatively rare elements that together account for only 0.5 per cent of the lithosphere. It is a metal belonging to the oxygen group of the periodic system, which also includes molybdenum, tungsten, and uranium.

The properties of chromium render it exceedingly valuable for many uses, the metal and its alloys being tough and resistant to corrosion. Its oxide is extensively used as a refractory.

Metallic chromium is bluish white in color. It possesses high metallic luster and takes a brilliant polish; it crystallizes in the cubic system, the crystalline metal being relatively brittle. Chromium has a hardness of about 9, approximately that of sapphire; its scratch hardness is 2,000, compared with 1,950 for case-hardened steel, 750 for ordinary steel shafting, and 408 for Swedish iron. The specific gravity of chromium at 20°C. is 6.92, and its atomic weight is 52. It melts at 1,520°C., boils at 2,200°C., and its specific heat is 0.1039 at 0°C. and 0.1872 at 600°C.

Chromium dissolves slowly in cold hydrochloric, dilute nitric, and dilute sulphuric acids, but dissolves rapidly in hot hydrochloric acid and concentrated sulphuric acid. It is unaffected by hot concentrated nitric acid. The pure metal is unaffected by air, oxygen, or chlorine at temperatures up to 300°C. but oxidizes readily at temperatures in excess of 1,200°C.; it is particularly useful for its resistance to corrosion from the atmosphere, rain, or sea water and from sulphur compounds such as hydrogen sulphide and others existing in petroleum and in rubber dough. It is also of importance industrially because of its resistance to corrosion from molten zinc, tin, brass, ammonia, lactic and acetic acids, and other industrial organic acids except oxalic acid.

The soluble chromium compounds are very poisonous.

OCCURRENCE

Native chromium has not been found, but a number of minerals found in nature contain appreciable amounts of chromium. The following is a list of the more common chromium-bearing minerals:

Commonly occurring chromium-bearing minerals

Mineral	Molecular composition	Chromium content	
		Cr	Equivalent Cr_2O_3
Chromite	$FeO.Cr_2O_3$	46.5	68.0
Daubreelite	$FeS.Cr_2S_3$	36.3	53.1
Uvarovite	$3CaO.Cr_2O_3.3SiO_2$	20.9	30.6
Crocoite	$PbCrO_4$	16.1	23.5
Stichtite	$7MgO.Cr_2O_3.2CO_2.8H_2O$	15.6	22.8
Phoenicochroite	$3PbO.2CrO_3$	12.0	17.5
Bellite	$PbCrO_4.XAs_2O_3$	1/ 11.8	17.3
Dietzeite	$7Ca(IO_3)_2.8CaCrO_4$	1/ 10.5	15.3
Vauquelinite	$2(Pb,Cu)CrO_4.(Pb,Cu)_3P_2O_8$	1/ 7.3	10.7
Redingtonite	Hydrous Cr sulphate	1/ 5.1	7.5
Knoxvillite	Hydrous basic Fe,+++ Al-Cr sulphate	1/ 5.1	7.4

1/ Variable composition; typical analysis given.

Of all the minerals containing chromium, chromite is the only one of commercial importance as an ore of chromium.

PROPERTIES AND OCCURRENCE OF CHROMITE

Chromite ($Cr_2O_3.FeO$) has a theoretical composition of 68 per cent of chromium sesquioxide (Cr_2O_3) and 32 per cent of ferrous oxide (FeO), but in the mineral, as found in nature, some of the iron has usually been replaced by varying amounts of magnesium, or some of the chromium has been replaced by alumina, or ferric iron (Fe_2O_3), or both.

Chromite is usually found in ultra-basic igneous rocks or their metamorphic equivalents such as serpentine - the most common gangue rock. It occurs disseminated or in compact masses, lenses, or stringers. Its origin has generally been ascribed to magmatic segregation, but recent studies of Ross3/ and Sampson4/ indicate that some chromite may not be due to this, but may have been introduced after solidification of the rock magma or during its later stages.

Chromite is black to brownish, sometimes yellowish red, in color; it has a brown streak, is translucent to opaque, has a submetallic lustre, and

3/ Ross, Clarence S., Is Chromite always a Magmatic Segregation Product: Econ. Geol., vol. 24, No. 6, Sept.-Oct., 1929, pp. 641-645.
4/ Sampson, Edward, May Chromite Crystalize Late: Econ. Geol., vol. 24, No. 6, Sept.-Oct., 1929, pp. 632-641.

is sometimes feebly magnetic. The hardness of chromite is 5.5, its specific gravity 4.32 to 4.57; it is brittle and has an uneven fracture. Chromite is infusible, and gives a definitely green borax-bead test in both the oxidizing and reducing flames. In salt of phosphorus in the oxidizing flame, it gives a yellow color when hot, but on cooling changes to a fine emerald-green tint. With soda and nitre on platinum it fuses to a magnetic mass, which is yellow in color when cool. A sodium phosphate bead is green, whereas sodium carbonate gives a yellow opaque bead in the oxidizing flame. Chromite is insoluble in acids, but is decomposed by sodium carbonate with the formation of water-soluble sodium chromate.

The usability of chromite is usually determined by the ratio between iron and chromium in the ore. Iron increases the fusibility of chromite, making it less desirable for refractories, whereas a high iron content renders chromite less usable for making ferrochromium. The usual ratio of chromium to iron in commercial chromite is about 2-1/2 : 1. The ratios of chromium to alumina, to magnesia, and to silica are important also because these constituents increase the amount of slag made in ferrochromium smelting and thus increase the metallurgical loss of chromium. Silica is usually limited to 5 per cent, as it is undesirable in ferrochromium except for making a few special alloys. Much of the better foreign ore contains from 48 up to 55 per cent of chromium sesquioxide, but the average domestic ore has a content of only about 40 per cent and its iron content is relatively high. Concentration by gravity of domestic ores of high iron content usually increases the iron content as well as that of chromium, so that the ratio is little improved. Foreign chromite is therefore chiefly in demand and there is little market demand for domestic chromite. The following table gives representative analyses of marketable ores from various countries:

Representative analyses of chromite from various countries[1/]

Country of origin	Percentage of principal constituents					
	Chromic oxide (Cr_2O_3)	Iron oxide (FeO)	Magnesium oxide (MgO)	Aluminum oxide (Al_2O_3)	Silicon dioxide (SiO_2)	Calcium oxide (CaO)
India:						
Baluchistan	54.00	13.60	16.40	11.32	1.86	1.10
Mysore	51.04	22.50	12.00	7.53	4.50	.60
Rhodesia: Selukwee	54.40	14.45	11.70	11.06	6.00	1.00
Canada	46.00	22.50	4.90	8.09	7.70	5.60
Russia	50.80	21.60	13.00	3.30	5.40	.60
Asia Minor	52.90	15.70	16.40	6.30	1.10	--
Hungary	39.00	16.10	17.20	17.50	8.00	--
United States: California	44.70	14.00	16.50	16.00	8.00	--
Greece	41.66	14.70	16.66	18.24	2.00	5.76
New Caledonia	55.02	12.77	8.00	11.10	3.00	.50

1/ Hoar, H. M., World Trade in Chromite: Supplement to Commerce Reports, Trade Information Bull. 252, p. 9.

Certain iron ores of Cuba, Celebes, the Gold Coast, and Greece contain from less than 1 per cent to 3 per cent of chromium associated with a small percentage of nickel; they are classed as chromiferous iron ores.

USES

The world's production of chromite finds its way into metallurgical, refractory, or chemical uses. It has been estimated that the consumption in the United States in 1927 was divided as follows:[5] Metallurgical, 46 per cent; refractory, 41 per cent; chemical, 13 per cent.

Metallurgical Uses

Preparation of Metallic Chromium

Metallic chromium may be prepared in the following ways:[6]

1. Reduction of Cr_2O_3 with carbon in the electric furnace.

2. Reduction of Cr_2O_3 with magnesium or aluminum by the thermic process.

3. Reduction of chromic chloride, $CrCl_3$, with potassium, sodium, or zinc.

4. By electrolysis of solutions of chromium compounds, usually chromic acid.

Metallic chromium is used to some extent in the manufacture of high chromium content alloys. It is also used as a protective coating on metals where it is deposited by electrolysis.

Ferrochromium

Ferrochromium is an alloy of iron, chromium, and carbon and is the basic alloy used in the manufacture of most chrome alloys. For the manufacture of ferrochrome chromite, mixed with anthracite coal, is reduced in the electric furnace[7] at a temperature of about 1,185° C. Reduction takes place theoretically according to the following formula:

$$FeO + Cr_2O_3 + 4C = Fe.2Cr + 4CO.$$

The resulting alloy usually contains 60 to 70 per cent of chromium, 25 to 32 per cent of iron and carbon in three ranges; 4 to 8 per cent, 1 to 2 per cent, and under 1 per cent. The amount of silicon is generally specified by the consumer; it varies according to the use for which the ferrochromium is intended. Low carbon ferrochromium is made by retreating high carbon ferrochromium with lime slag, chromite ore, and fluorspar. Owing to metallurgical

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5/ Sampson, Edward, Chromite: Eng. and Min. Jour., vol. 125, Jan. 21, 1928, p. 104.
6/ Friend, J. H., Textbook of Inorganic Chemistry: Vol. 7, pt. 3, p. 9.
7/ Lyon, D. A., Keeney, R. M., and Cullen, J. F., The Electric Furnace in Metallurgical Work: Bull. 77, Bureau of Mines, 1914, pp. 131-141.

difficulties encountered in this retreatment, the price for low carbon ferrochromium is generally 2 to 3 times as much as for high carbon ferrochromium.

Ores used in the manufacture of ferrochromium usually contain 45 to 50 per cent of chromium sesquioxide and less than 5 per cent of silica. Silica is undesirable because the chemical reactions of silicon and carbon are so similar that there is great difficulty in separating them metallurgically. A typical ore used in the manufacture of ferrochromium contains at least 40 per cent of chromium sesquioxide, a maximum of 5 per cent of silica, a maximum of 0.5 per cent of sulphur, and a maximum of 0.2 per cent of phosphorus. Some operators claim that a 60 per cent ferrochromium can be manufactured from ores containing 30 per cent of Cr_2O_3, 10 per cent of FeO, and traces of sulphur and phosphorus. Lump ore is preferred over concentrates or fine ore, since with the latter there is said to be a high loss of chromium, perhaps, amounting to as much as 10 per cent. Concentrates generally have high chromium content, but at the same time they are often high in iron, and since nearly all of the iron finds its way into the alloy, it is difficult to obtain the 60 or 70 per cent of ferrochromium usually required. High magnesia and alumina increase the cost of manufacture because they increase the amount of slag to be handled.

Chromium Alloys

Chromium is an essential constituent of many ferrous and nonferrous alloys, for which use its hardening and corrosion-resisting properties are of great value. These alloys are finding increased application in many branches of industry.

The properties and uses of chromium alloys are affected by several factors, such as carbon content, heat treatment, and the content of other alloying metals--nickel, manganese, silicon, vanadium, etc.--; however, a discussion of this phase of the metallurgy of chromium is too involved for detailed presentation in this paper. An excellent general discussion of chromium alloys was presented by Frederick M. Becket in Mining and Metallurgy of December, 1928, and January, 1929.

Becket classifies the chromium steels into three groups, as follows:

1. Low-chromium steels containing 5 per cent or less of chromium.

2. Medium-chromium steels containing up to 18 or 19 per cent of chromium.

3. High-chromium steels containing still higher percentages of chromium.

Most of the low-chromium steels contain about 1 per cent of chromium and are useful for their strength, toughness, and hardness. These types of steel are used in antifriction bearings, case-carburizing steels employed in gears, etc., cutlery and small tools requiring maintenance of a keen

edge, and crusher parts and other articles requiring strength and resistance to abrasion. The addition of nickel induces greater toughness and dynamic strength without the loss of hardness. Small amounts of vanadium produce high tensile strength and fatigue-resisting properties; these steels are used in the axles, crankshafts, etc., of motor cars. Molybdenum also improves the fatigue resistance of low-chromium steels and produces better machinability, weldability, and wider latitude in heat treatment. Some high-speed steels contain from 3-1/2 to 5 per cent of chromium and 11.5 to 18 per cent of tungsten.

Valve steels containing 7 to 9 per cent of chromium and 2.5 to 4 per cent of silicon are employed in the manufacture of valves used in internal combustion engines. The so-called stainless steels used in cutlery contain from 12 to 14 per cent of chromium and 0.25 to 0.40 per cent of carbon. The addition of nickel increases the corrosion resistance of chromium steels. Steels containing about 18 per cent of chromium and 8 per cent of nickel are widely used in restaurant equipment, automobile parts, chemical equipment, etc., on account of their resistance to oxidation and chemical corrosion. Rustless irons containing 12 to 18 per cent of chromium and carbon below 0.10 per cent are more readily worked than stainless steels and are used in rust-resisting parts requiring considerable fabrication.

High-chromium steels and irons are used chiefly where resistance to oxidation and corrosion at high temperatures is required, such as in heat-treating furnaces, annealing furnaces, and roasting furnaces. High-chrome alloys containing 50 per cent or more of nickel are extremely resistant to oxidation at high temperatures.

The nonferrous alloys of chromium, nickel, and tungsten or chromium, cobalt, and tungsten (with variable amounts of molybdenum or vanadium) form highly durable and wear-resisting materials, which can be welded onto tools, excavating machinery, drill bits, etc., to build up the worn parts. They are also used in the making of precision and surgical instruments, for decorative purposes, etc.

Ferrochromium is the principal source of chromium used in the manufacture of chromium alloys, but in some of the nonferrous alloys use is made of metallic chromium produced in the electric furnace or by the thermit process.

Refractories

The second largest use of chromite is in the manufacture of refractory materials such as brick and cement. Its use here depends upon its chemical stability and its neutral character, which permits its use as a parting between silica and magnesite bricks. Chromite refractories are very resistant to corrosion by metals, slags, and vapors. They have a fairly high melting point, relatively high thermal conductivity, and relatively low electric resistivity.

The ratios of chromium to iron, alumina, and silica in chromite ores are important in the manufacture of refractories. The normal ratio of Cr_2O_3 to iron varies from 2 : 1 to 3 : 1; when the iron content increases in respect to the chromium content, the ore becomes less usable on account of its increased fusibility. The ores most generally used in the manufacture of refractories contain from 38 to 48 per cent of chromium sesquioxide, 12 to 24 per cent of alumina, 12 to 24 per cent of iron, 14 to 18 per cent of magnesia, and up to 10 per cent of silica. Most manufacturers prefer ore containing 4 to 5 per cent of silica, but some ore containing as high as 10 per cent of silica is used. Lump ore is preferred by manufacturers of chromite bricks. Greece, Cuba and the Union of South Africa are the principal sources of refractory-grade chromite.

Most of the chromite used as a refractory is manufactured into bricks. Chromite bricks fuse at 1,950 to 2,050°C.; the melting point is usually given at 2,050°C. They have a specific gravity of 3.9 to 4, and their coefficient of thermal expansion is fairly high. Chrome bricks have great strength at low temperatures but are apt to fail suddenly at about 1,450°C. Their principal competitor is magnesite bricks which have a slightly higher melting point (2,165°C) and greater resistance to abrasion but less resistance to corrosion than chromite bricks. In recent years chromite bricks have replaced magnesite bricks to a large extent, partly because of the increased cost of magnesite bricks. In 1923 it was estimated that approximately three magnesite bricks were manufactured for every chromite brick, but in 1927 it was estimated that these two refractories were manufactured in about equal amounts. Although there is no exact information available, it appears that this trend has continued since 1927.

Chromite bricks have fair resistance to abrasion but are said to be objectionable for furnace bottoms because of their inability to withstand the severe mechanical erosion encountered there. They are well suited for lining along the slag line of basic open hearth steel furnaces, in which a magnesite refractory bottom and silica brick sides and roof are used. This application is based on the ability of chromite to withstand the corrosive action of slag, metals, and gas. Magnesite is preferred for the bottom of the furnace because of its greater resistance to mechanical erosion. Chromite bricks were substituted for silica bricks in reverberatory furnaces used in copper refining and showed satisfactory resistance to corrosion and spalling, but owing to the large amount of metal absorbed by the chromite, which could only be recovered at considerable expense, this use was found to be uneconomical. Among the more recent applications of chromite brick is its use as a lining for kilns used for drying sulphite paper pulp. This application is based upon the resistance of chromite to corrosion from the sulphurous gases liberated in the kilns.

Crushed chromite and chromite cements are also used as refractory materials.

Chemical Uses

About 13 per cent of the chromite consumed in the United States is manufactured into chemicals for the electroplating, dyeing, tanning and pigment industries. Ore containing about 47 per cent or more of Cr_2O_3 and free of sulphur is usually desired for this purpose. Some ore of lower chromium content enters the chemical industry, but presumably it is mixed with higher-grade material. The silica content should be under 8 per cent and preferably under 5 per cent. High silica results in the formation of excessive amounts of sodium silicate which is a waste product in the manufacture of some chromium salts. The use of low-grade ores is objectionable because of increased bulk of ore to be handled, lower furnace efficiencies, and increased loss of other chemical reagents due to combination with impurities. Fine ore is slightly more desirable than massive ore on account of the slight saving in crushing, and for this reason fine concentrates of chromite are acceptable to the chemical industry.

The following list contains the principal industrial chromium chemicals:

Sodium bichromate ($Na_2Cr_2O_7$) and potassium bichromate ($K_2Cr_2O_7$) are principal commercial salts of chromium. They are used in the manufacture of many other chromium salts and pigments, as a discharge in bleaching oils and fats, in chrome tannage, as mordants in the dyeing industry, and as an oxidizing agent. Owing to its lower cost and greater solubility, sodium bichromate is more widely used than potassium bichromate.

Sodium chromate (Na_2CrO_4) and potassium chromate (K_2CrO_4). These salts are intermediate products in the manufacture of most chromium compounds. Sodium chromate results from the roasting of a mixture of chromite ore, sodium carbonate, and lime. It is extracted from the calcine by leaching with hot water. After purification of this solution the sodium chromate may be recovered by crystallization or, by the addition of sulphuric acid to the solution, sodium bichromate is formed which may be recovered by crystallization. The chromates are used largely in the manufacture of other chemicals, in dyeing, and in treating feed water for the prevention of boiler scale.

Chromic anhydride (CrO_3) is sometimes referred to as chromic acid. It is the principal constituent of chromium electroplating baths and a powerful oxidizing agent.

Chromium sesquioxide (Cr_2O_3). This compound is the only commercial ore of chromite but is rarely found sufficiently pure for chemical uses. The pure oxide can be prepared by reducing a bichromate with sulphur or starch. It is used in the manufacture of chromium metal and as a pigment.

Chromium sulphate ($Cr_2(SO_4)_3$) is an excellent mordant and is sometimes used in electroplating baths. Basic chromium sulphate is used in the tanning industry.

Chromium acetate ($Cr(C_2H_3O_2)$ $3H_2O$) is used mainly as a mordant and as a dye for olive khaki shades. It is marketed as a heavy solution containing about 10 per cent of Cr_2O_3.

Chromium chloride ($CrCl_3$) is used as a mordant in calico printing and to a limited extent in the tanning industry.

Chromium alum ($Cr_2(SO_4)_3.K_2SO_4.24H_2O$) is usually obtained as a by-product in operations where a mixture of potassium bichromate and sulphuric acid acts as an oxidizing agent. It is used as a mordant in the dye industry.

The following list contains the principal chrome pigments:

Chrome yellows are compounds of lead with chromic acid, which range in color from deep orange to light yellow. "Chrome yellow medium" is a pure lead chromate, while the lighter shades are mixtures of lead chromate and lead sulphate or other insoluble lead salts. The orange shades are made of basic lead chromate. Chrome yellows have brilliant color, good color strength, and good hiding power but are only fairly durable and turn black in contact with sulphur.

Zinc chromate and barium chromate are of light yellow color but have a limited use due to their high cost and low hiding power and color strength. Zinc chromate is soluble in water and can only be used in interior paints.

Chrome greens are mixture of Prussian blue and chrome yellow in various proportions according to tone. They have great hiding power and good color strength but are not very durable.

Chrome oxide green is made of chromic oxide (Cr_2O_3). It is a rather expensive, permanent, dull color of satisfactory hiding power. A carefully made product of the best grade is known as "Guignets green" or "viridian."

Zinc green is a mixture of zinc chromate and Prussian blue. It has a brilliant color and is permanent to light but not permanent to water and alkali.

Chrome red (American vermillion) is a basic chromate of lead. It is a brilliant scarlet, coarsely crystalline powder, with good hiding power and color strength. As a protective pigment in paints for iron and steel it equals or excels red lead. Its use is limited because of its high price.

Chrome Plating

The chromium plating industry has developed very rapidly in the last few years, owing to three principal characteristics of chromium plate. First, and perhaps the most important, is the brilliant lustre of chromium and the readiness with which it is brightly polished; second, the resistance of chromium to corrosion; and third, its hardness and resistance to abrasion by sliding or rubbing wear. The brilliant lustre of chromium has led to its use

for many ornamental purposes such as decorative parts on automobiles, decorative parts of buildings, and household articles, such as trays, cutlery, mirror frames, plumbing fixtures, etc. The resistance of chromium plating to corrosion has extended its use into cooking utensils and other kitchen equipment, oil-cracking plants, sulphite paper pulp mills, and other industries where corrosive gases and high temperatures are encountered. In this field of application, chromium plate meets competition from chromium steels and chromium nickel steels.

The hardness of chromium has led to its use in plating and engraving plates, dyes, precision instruments, etc. Chromium plating is especially valuable on tapes, linear scales, verniers, micrometers, etc., due to its hardness, which protects the etched markings, and to the fact that when worn the plate can be removed and a new one deposited without obliterating the original markings. Owing to its hardness and its ability to take a high polish, chromium plating is employed in certain types of rolls used in crushing soft materials. Chromium-plated cutting tools are superior to ordinary steel tools for cutting slate, asbestos board, etc. Chromium plated saws, chisels, lathe tools, gears, etc. have been used successfully.

During the early stages of the industry, a large volume of inferior chromium-plated goods was placed on the market, the performance of which was so poor as to create public prejudice. However, constant improvement in technology has resulted in a much better product and public confidence is steadily being restored. At present there is a tendency to substitute chrome-nickel stainless steels for chromium-plated articles, but the ultimate outcome of this tendency is not yet manifested.

Recent papers by Schneidewind[8], Haring and Barrows[9], and Farber[10], give details of chromium-plating technology.

MARKETING CHROMITE

Trade Requirements

The requirements of the various industries using chromite have been discussed in detail under uses of chromite. They are briefly summarized as follows:[11]

8/ Schneidewind, Richard, A Study of Chromium Plating: Eng. Research Bull. 10, Univ. of Michigan, Ann Arbor, Mich., 1928, Price $1.

9/ Haring, H. E., and Barrows, W. P., Electrodeposition of Chromium from Chromic-Acid Baths: Tech. Paper 346, Bureau of Standards, Dept. of Commerce, Washington, D. C., 1927, 39 pp.

10/ Farber, H. S. The Throwing Power on Chromium Plating: Metal Ind., vol. 28, No. 3, March, 1930, pages 124-125.

11/ Engineering and Mining Journal, Marketing of Chrome Ore: vol. 125, No. 12, Mar. 24, 1928, p. 501.

Most buyers of chrome ore are satisfied if the chrome and iron contents are stated, though sometimes the percentages of silica or alumina are also desired. Buyers for the steel or ferro-alloy industries usually desire an ore containing not less than 45 per cent of chromic oxide and an iron content not exceeding one-quarter of the combined iron and chromic oxide. For instance, if the chromic oxide content is 45 per cent, the iron must not exceed 15 per cent. The chemical trade requires the chromic acid content to be 50 per cent or better, but is not so particular about the iron content, and the limit already given might be exceeded by 1 or 2 per cent. Manufacturers of refractories demand low silica, generally below 5 to 8 per cent.

Ore of low chromic acid content is acceptable to the ceramic and brick industry, though, of course, it does not command so high a price. As low as 38 per cent of chromic aci is quite commonly accepted, and business has been done in even lower grades when the shipping facilities from mine to consumer were particularly advantageous, so that an unusually low price could be quoted.

No recognized standard exists for premiums and penalties. In sales to the steel and chemical trades it is customary to quote a price per ton based on a definite percentage of chromic oxide - say, 20 dollars for 50 per cent material with an allowance of 40 cents for each unit above or below this analysis. The brick and refractory trades, however, buy ordinarily as is, with a guaranteed minimum content of chromic acid and with no premium for a better quality than is specified.

Prices

The price of chromite varies according to its content of chromium sesquioxide, as is illustrated by the following quotations taken from Metal and Mineral Markets of December 4, 1930: Indian, Rhodesian, and Russian ore containing 46 to 48 per cent of Cr_2O_3, \$19.50 per long ton, f.o.b. Atlantic ports; and Rhodesian and New Caledonian ore containing 50 to 51 per cent of Cr_2O_3, \$24. The former was at the rate of 42 cents per unit of Cr_2O_3 and the latter was 47-1/2 cents.

The following table gives the approximate average prices per long ton of high-grade ore in the United States from 1914 to 1930:

Approximate average price of chromite in the United States, f.o.b. Atlantic ports

Year	Average price per long ton	Year	Average price per long ton
1914	$14.75	1923	$19.00
1915	11.20	1924	21.35
1916	15.44	1925	21.95
1917	24.00	1926	22.46
1918	44.99	1927	22.88
1919	17.25	1928	1/ 23.00
1920	17.93	1929	1/ 22.00
1921	10.28	1930	1/ 21.50
1922	22.53		

1/ Quotations vary by different grades of ore and by periods, making average price determinations very difficult to obtain. The 1929 prices range from $21.50 to $25, while those of 1930 range from $19.50 to $24.

The following table shows the average London quotations per long ton for various grades of ores, from 1926 to 1930:

Average London prices for chromite per long ton, 1926 to 1930, c.i.f. British ports

Type of ore	Cr_2O_3 content, per cent	Quotations per long ton				
		1926	1927	1928 1/	1929 1/	1930 1/
Rhodesian low-grade	45 to 47	$20.70	$21.27	$20.41	$21.79	$20.00
Indian medium-grade	47 to 48	21.44	22.03	21.65	22.60	20.00
Rhodesian and Indian high-grade	48 to 52	22.30	22.36	22.90	23.90	24.00
New Caledonian high-grade	50	25.00	25.00	25.00	25.00	25.00

1/ Estimated.

DOMESTIC TARIFFS

Chromite ore was subject to a duty of 15 per cent from 1883 until 1894, when it was put on a free list, where it has remained under succeeding tariff laws. Although under the tariff act of 1922, chromite or chromite ores were free of duty, under paragraph 72 of the same act, "chrome yellows, chrome greens, and other colors containing chrome in pulp, dry or ground in or mixed with oil or water," were subject to a duty of 25 per centum ad valorem.

The tariff act of 1930[12] does not change the duty on pigments or ores, but places a duty of 2-1/4 cents per pound on potassium chromates and bichromates, and 1-3/4 cents per pound on sodium bichromates and chromates. The duty on ferrochromium containing 3 per cent or more of carbon amounts to 2-1/2 cents per pound of chromium content; on ferrochromium containing less than 3 per cent of carbon and on chromium metal the duty is 30 per cent ad valorem (par. 302 (k)). According to paragraph 305, page 25 of the act, an additional ad valorem duty of 8 per cent is charged on iron and steel products (described in pars. 303, 304, 307, 308, 312, 313, 315-319, 322-324, 327, and 328) which contain in excess of two-tenths of 1 per cent of chromium.

WORLD TRADE IN 1928

In reviewing the figures for the flow of chromite in 1928, it is evident that the six main producing countries consume little chromite and that the main consuming countries, excepting Russia, produce only negligible amounts. These facts illustrate how an important manufacturing industry may depend largely upon imported raw materials.

The production of chromite in 1928 amounted to about 450,400 tons, of which nearly 87 per cent was produced by Rhodesia, New Caledonia, British India, Cuba, the Union of South Africa and Russia. The world total exports amounted to about 364,000 long tons, 89 per cent of which was exported by the above countries. The total imports amounted to around 350,000 tons, of which the United States, Germany, Sweden, the United Kingdom, Norway, and France took 95 per cent. These same countries accounted for over 84 per cent of the total world consumption of chromite during the year. Russia, Yugoslavia, and Japan were the only large consuming nations which were able to produce sufficient chromite to meet their needs. Table 1 shows the amount of chromite produced, exported, imported, and consumed by the principal countries in 1928, in long tons.

12/ The Tariff Act of 1930: Document No. 196, 71st Cong., 2nd Sess., Washington, D. C., pars. 70, 78, and 81, pp. 12-13.

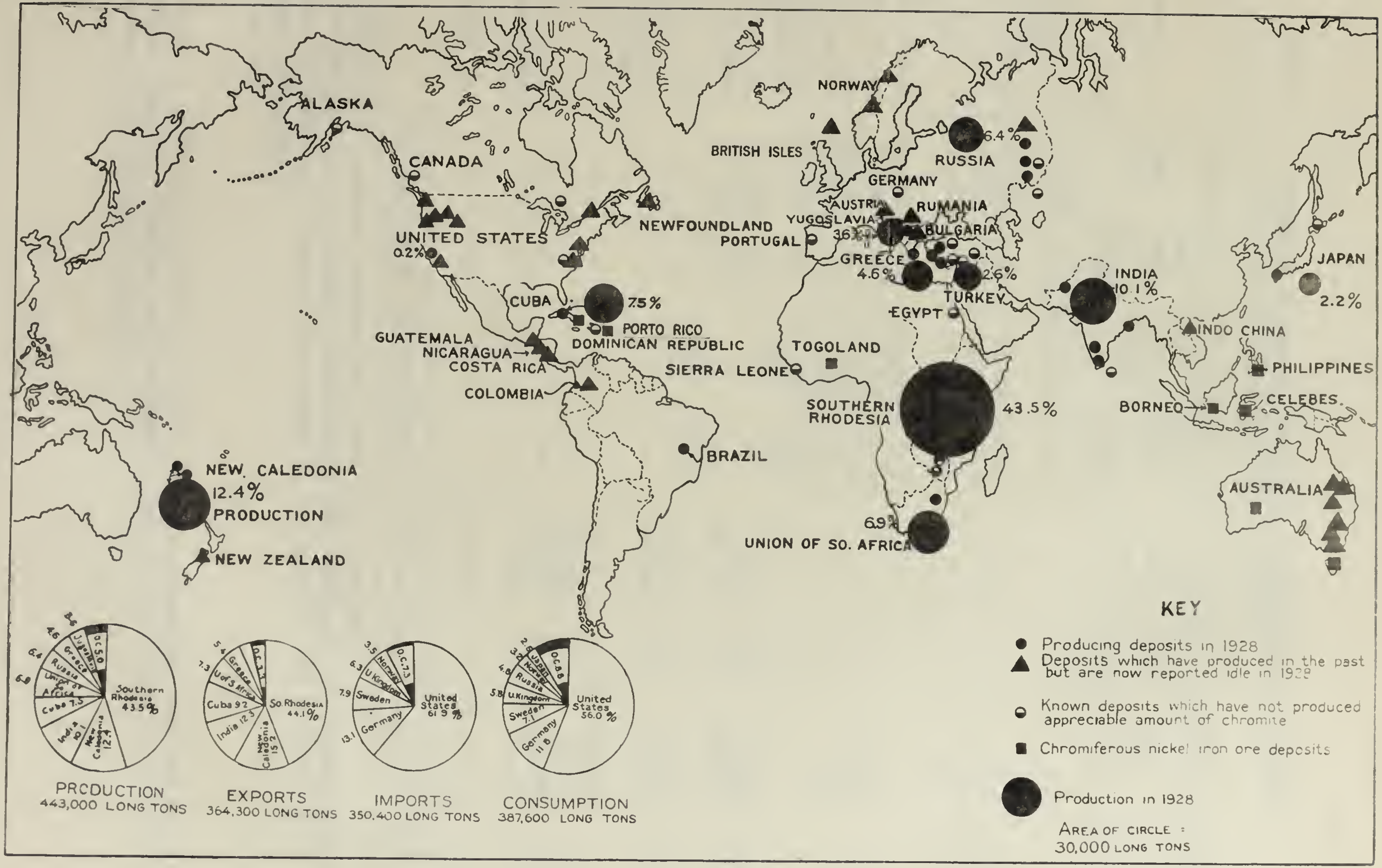

Figure 1.—Location of world chromite deposits; and principal relative production, exports, imports, and consumption, 1928

Table 1. - Chromite produced, exported, imported, and consumed by the principal countries in 1938

Country	Production: Long tons	Production: Per cent of total	Exports: Long tons	Exports: Per cent of total	Imports: Long tons	Imports: Per cent of total	Apparent consumption: Long tons 1/	Apparent consumption: Per cent of total
Southern Rhodesia ...	195,900	43.5	162,300	44.1	--	--	--	--
New Caledonia	56,000	12.4	56,000	15.2	--	--	--	--
India	45,400	10.1	45,300	12.3	--	--	200	0.1
Cuba	33,700	7.5	33,700	9.2	--	--	--	--
Union of South Africa	31,300	6.9	23,400	7.3	--	--	--	--
Russia	29,000	6.4	3,000	0.8	--	--	26,000	6.6
Greece	20,600	4.6	19,800	5.4	--	--	800	0.2
Yugoslavia ..	16,400	3.6	9,100	2.5	--	--	7,300	1.8
Turkey	11,700	2.6	11,700	3.2	--	--	--	--
Australia ...	--	--	(2/)	(2/)	--	--	--	--
Japan	9,700	2.2	--	--	(3/)	(3/)	9,800	2.5
United States	700	0.2	--	--	216,600	61.9	217,300	55.0
Brazil	--	--	(2/)	(2/)	--	--	--	--
Germany	--	--	--	--	45,800	13.1	45,800	11.6
Sweden	--	--	--	--	27,600	7.9	27,600	7.0
United Kingdom	--	--	--	--	22,400	6.3	22,400	5.7
Norway	--	--	--	--	12,400	3.5	12,400	3.1
France	--	--	--	--	4/8,000	2.3	4/8,000	2.0
Czechoslovakia	--	--	--	--	2,400	0.7	2,400	0.6
Other countries ..	--	--	--	--	4/15,200	4.3	4/15,000	3.8
	450,400	100.0	364,300	100.0	350,400	100.0	395,000	100.0

1/ Producers and consumers stocks not taken into account.
2/ A few hundred tons of exports from Brazil and Australia is unaccounted for.
3/ A few hundred tons imported into Japan is included in the estimate for "Other countries."
4/ Estimated.

Figure 1 shows the location of world chromite deposits, and the relative production, exports, imports, and consumption of the principal producing and consuming countries during 1928.

WORLD PRODUCTION

The first mining of chromite is said to have occurred as early as 1820 at the Roros deposit in Norway, but the first well-established output came from the Reed Mine, in Maryland, about 1827. Thereafter the United States led the world in chromite production until 1860 when Turkey took the lead. Russia in turn became the chief producer from 1897 to 1902, inclusive. From 1903 to 1909, New Caledonia held first place, but alternated with Southern Rhodesia from 1910 to 1917, inclusive. In 1918, 1919, and 1920-1921, the United States, India, and New Caledonia led, respectively. In 1922 Southern Rhodesia assumed first place and has since continued to be the world's largest producer by a large margin. Figure 2 illustrates the world production of chromite from 1890 to 1929, inclusive.

Figure 3 shows the production of chromite from 1890 to 1929, inclusive, by the principal producing countries, in long tons.

Table 2 gives the production of chromite by countries for 1920 to 1929, inclusive, in long tons.

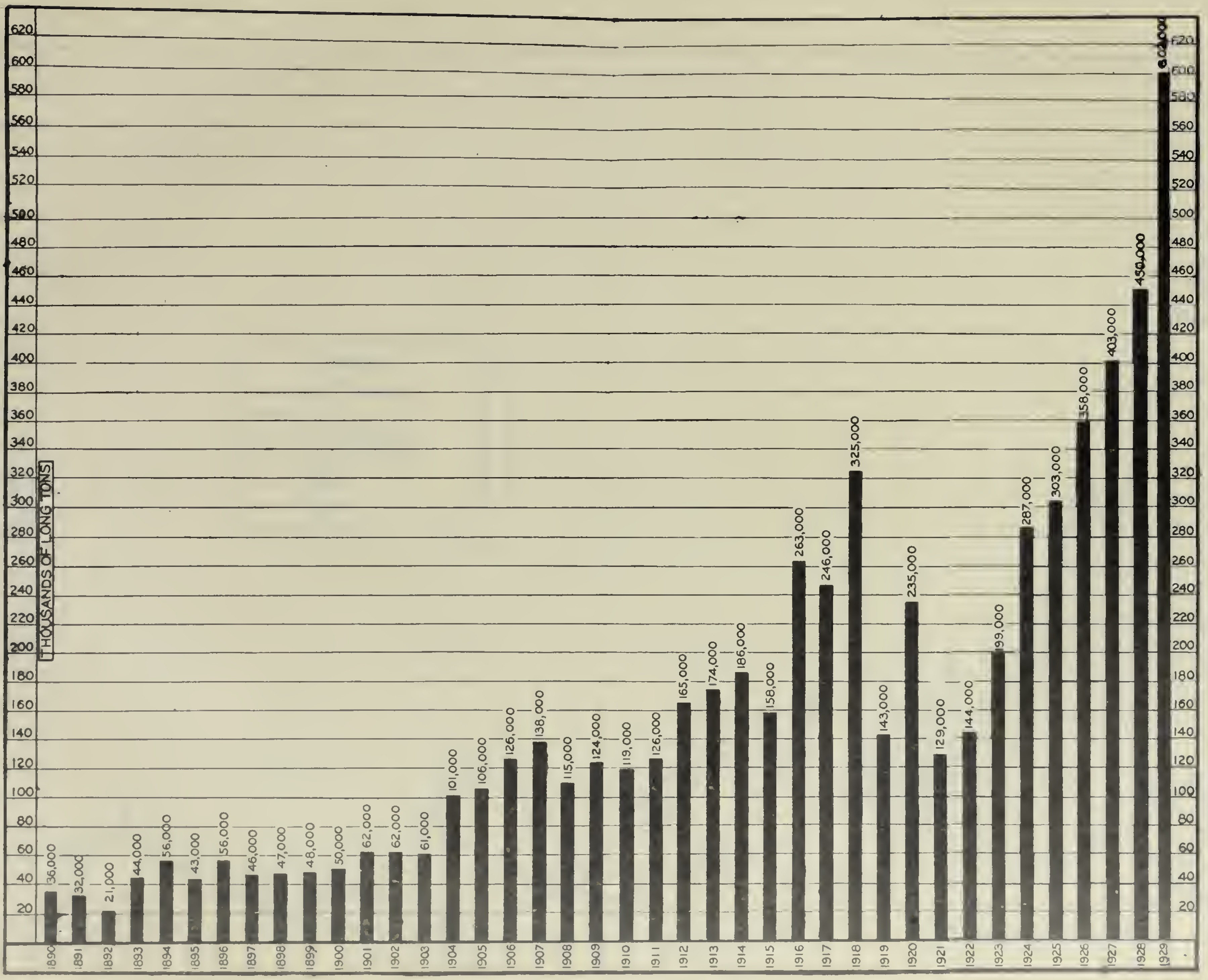

Figure 2.—World production of chromite, 1890–1929

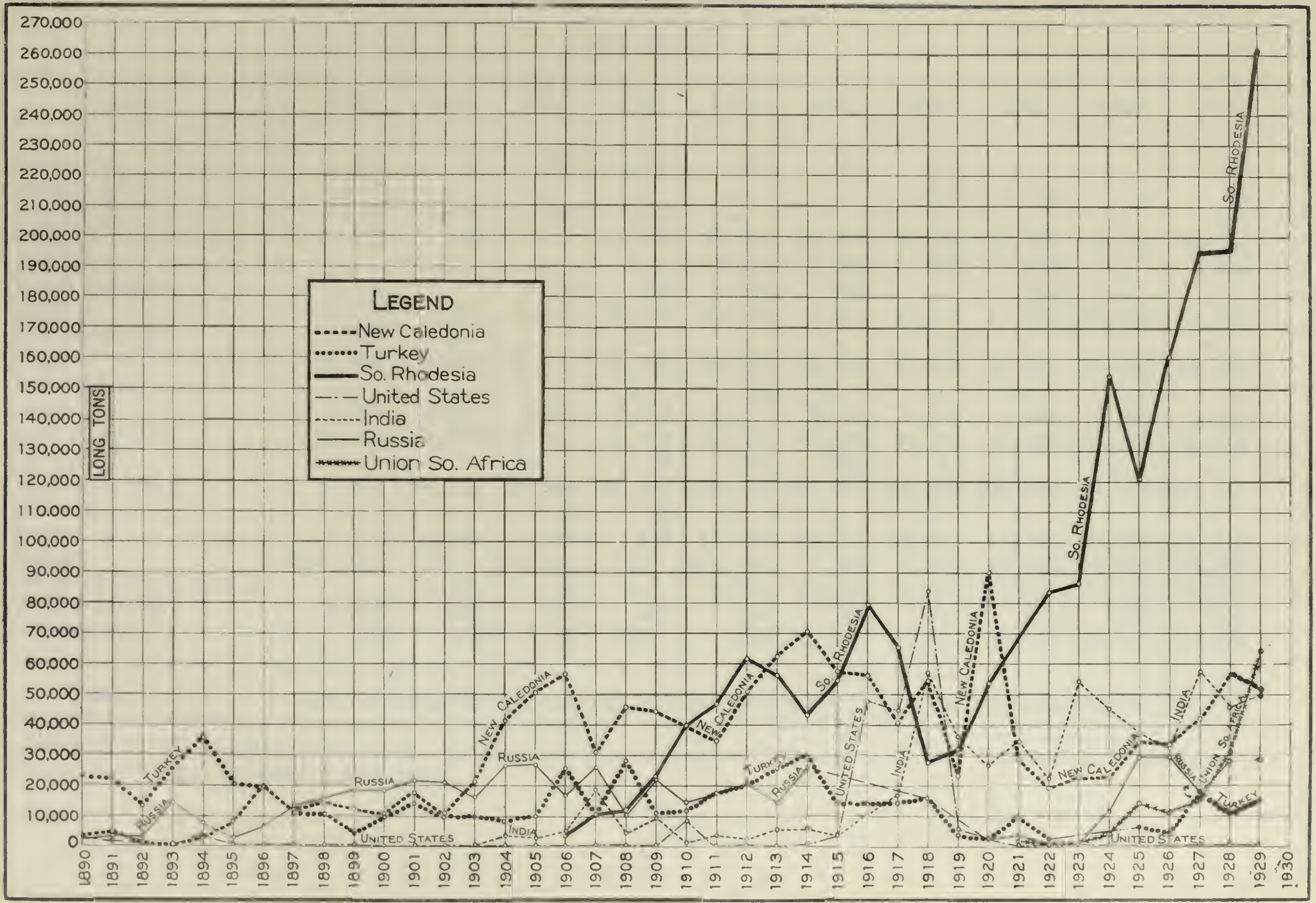

Figure 3.–Production of chromite in principal producing countries, 1890–1929

Table 2. -- World production of crude chromite, by countries, 1920 to 1929, inclusive

(Long tons)

Country	1920	1921	1922	1923	1924	1925	1926	1927	1928	1929
Australia	1,617	62	529	1,192	773	963	597	--	--	129
Brazil 1/	3,451	--	--	--	--	--	1,476	1,791	20	69
Canada 2/	9,835	2,498	685	3,177	--	--	--	--	--	112
Cuba 1/	710	600	--	10,420	19,567	29,830	36,020	16,984	33,707	52,950
Cyprus	--	--	--	--	2,811	1,989	640	712	--	2,444
Great Britain	1,100	--	595	546	1,043	448	--	378	--	--
Greece	12,295	7,902	9,068	14,594	14,420	7,951	19,732	17,041	20,622	23,832
Guatemala 2/	1,122	401	--	--	--	--	53	--	--	--
India	26,801	34,762	22,778	54,243	45,463	37,452	33,382	57,207	45,456	49,566
Indo-China	--	--	--	--	20	--	--	--	--	11
Japan	3,906	3,315	3,698	4,455	5,277	5,731	6,941	9,628	9,653	9,018
New Caledonia	90,089	28,993	19,068	22,853	22,657	34,184	33,726	42,208	56,004	51,763
Norway 1/	--	--	--	34	174	433	17	--	--	--
Rhodesia (So.)	53,811	44,810	83,460	86,317	154,219	121,275	161,780	194,658	195,916	261,710
Rumania	--	--	30	59	--	--	--	--	--	--
Russia 3/	2,918	3,950	834	878	11,706	29,636	29,883	18,989	29,070	29,525
Turkey	4/24,605	(5)	4/2,500	(5)	4/3,346	7,387	6,565	18,029	11,662	14,840
Union of South Africa	--	1,061	86	--	4,500	13,539	11,852	16,691	31,255	62,964
United States	2,502	282	355	227	288	108	141	201	660	269
Yugoslavia	10	10	15	(5)	295	11,968	15,731	8,619	16,415	42,343
Total	234,772	128,646	143,701	198,995	286,559	302,894	358,536	403,136	450,440	601,545

1/ Exports. 2/ Imports into U. S. 3/ Data for fiscal year ending Sept. 30. 4/ Estimated on basis of incomplete data. 5/ Data unavailable.

The total world production for the 103 years, 1827 to 1929, inclusive, is estimated at about 7,100,000 long tons, distributed as follows:

World production of chromite, 1827 to 1929, inclusive, by countries

Country	Production, long tons 1/	Per cent of total
Southern Rhodesia	1,912,000	26.9
New Caledonia	1,346,000	19.0
Turkey	832,000	11.7
Russia	650,000	9.2
India	616,000	8.7
United States	478,000	6.7
Greece	336,000	4.7
Cuba	213,000	3.0
Canada	175,000	2.5
Union of South Africa	139,000	2.0
Japan	112,000	1.6
Others	282,000	4.0
Total	7,091,000	100.0

1/ Figures are subject to revision at a later date.

WORLD RESOURCES

The world deposits of chromite were described in detail by Burchard and others in the 1918 volume of Mineral Resources of the United States. The following discussion gives a brief summary of the deposits in the United States and the principal producing countries.

United States

United States production of chromite from 1827 to 1929, inclusive, was 478,000 long tons, or 6.7 per cent of the world total.

Although chromite in the Maryland-Pennsylvania district was discovered about 1810,[13] mining did not begin until 1827. During the period from 1827 to 1860, this district led the world in production. It has been estimated that about 250,000 tons of ore[14] averaging about 40 to 63 per cent Cr_2O_3, was shipped from the Maryland-Pennsylvania district prior to 1882. In 1827 the Tyson Mining Co., through Isaac Tyson, began operations at the Reed mine, in the Soldier's Delight district, near Jarrottsville, Md. This ore was first used in the manufacture of yellow and green paints used for decorating carriages and furniture in Baltimore. Later a considerable amount was

13/ Baltimore Medical Philosophical Lyceum: Vol. 1, July 12, 1911, pp. 255-271.
14/ Furness, J. W., Chromite: Mineral Resources of the United States, 1925, Bureau of Mines, Pt. I, p. 141.

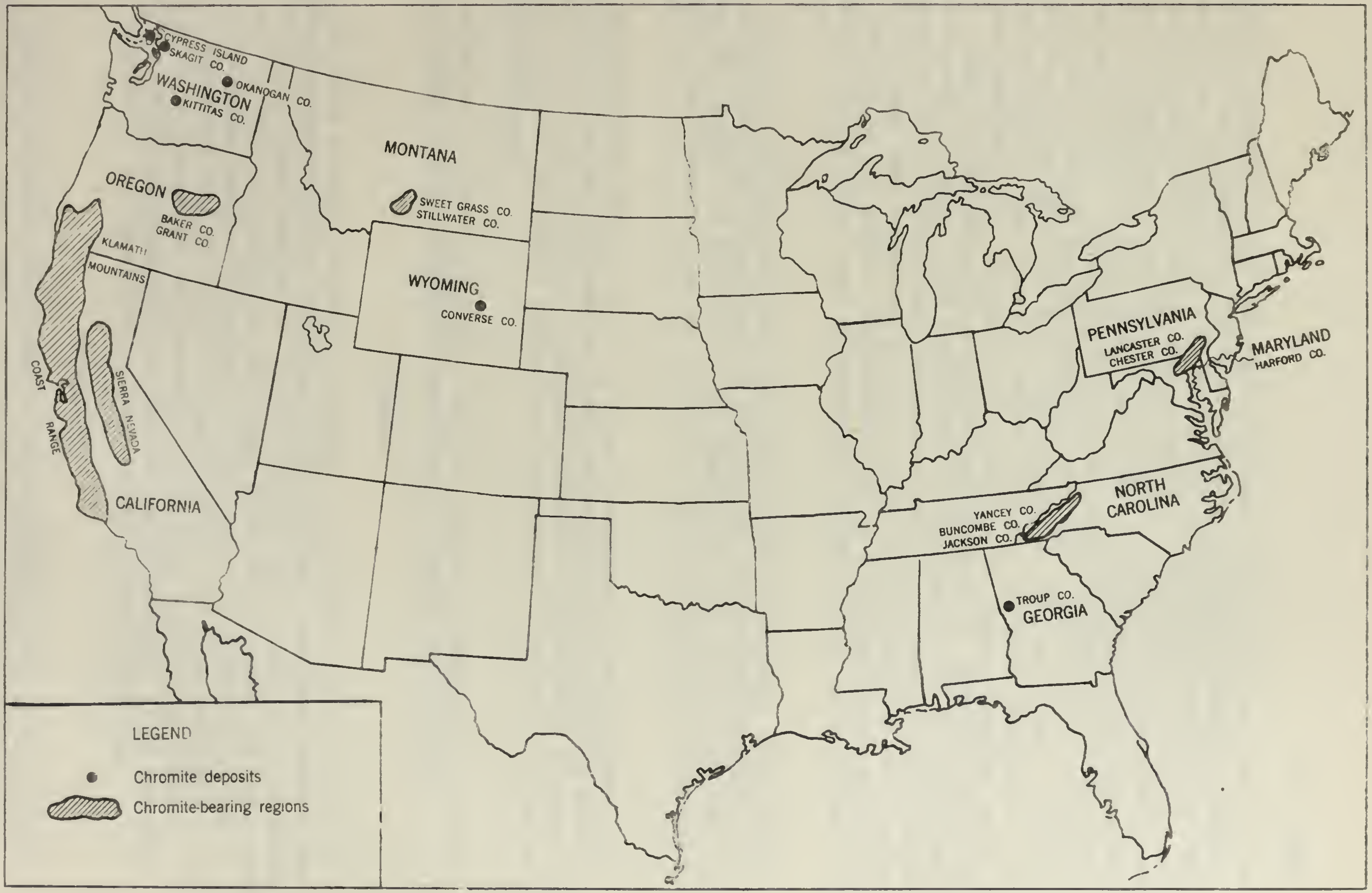

Figure 4.–Chromite deposits of the United States

exported to Great Britain and other European countries for the manufacture of pigments and chemicals, but when the larger and more lucrative Turkish deposits were developed, the foreign market for Maryland ores was lost and chromite mining in the United States declined. From 1870 to 1880 the yearly production decreased, but recuperated slightly with the discovery and development of chromite in California around 1880, culminating in a production of 3,680 tons in 1894. From 1900 to 1914 the annual average output was about 265 long tons, though imports steadily increased. The scarcity of shipping facilities during the World War caused the United States to turn to the deposits of Canada, Cuba, Brazil, Central America, and other countries of the Western Hemisphere for its supply of chromite. As these countries proved unable to supply requirements, an urgent demand arose for increased production from domestic deposits. The result was a production of 185,065 long tons of chromite in the United States from 1914 to 1919, inclusive, a peak production of 83,753 long tons being made in 1918. Most of this chromite was obtained from California and Oregon. Termination of the World War left United States producers with a large stock of high-cost chromite for which there was little demand.

In 1918 there were 450 shippers, distributed as follows: California, 374; Oregon, 60; North Carolina, 5; Montana, 3; Pennsylvania-Maryland, 3; Alaska, 3; Georgia, 1; Wyoming, 1; and Washington, 1. In 1919 and 1920 some chromite from stocks on hand at the end of 1918 was shipped from domestic mines.

As production during the war had been requested by the Government, a considerable sum was disbursed by a War Minerals Relief Commission in compensation for loss incurred by chromite producers. Since 1920, however, the yearly output of the United States has averaged about 230 tons and has never exceeded 660 tons.

Brief descriptions of the more important chromite areas of the United States, shown on the accompanying map (fig. 4), may be of interest.

Alaska

The total chromite reserves of Alaska, in 49 known deposits, have been estimated at about 227,900 short tons. The two most important deposits are located in the Kenai Peninsula. In 1917 mineral was extracted in the southwest end of the peninsula where ore bodies occur at tidewater and are reported to have reserves capable of yielding 33,000 tons of concentrates running from 40 to 45 per cent of Cr_2O_3. At Red Mountain, 16 miles northeast of the Kenai deposits, low-grade ores averaging between 30 and 40 per cent are found, but little development work has been done thus far. It is believed that the deposits are unprofitable under normal market conditions.

California

Chromite is widely distributed in the serpentine areas of the Sierra Nevada foothills, Coast Range, and Klamath Mountains, but most of the

deposits can not be mined profitably under normal market conditions. During the war there was considerable chromite mined, but with the collapse of the market after the war, production fell off to an insignificant amount. The total production to date has been about 290,000 long tons. Known reserves in 1925 were estimated at 220,000 tons which included about 35,000 tons of ore left in stocks at the end of the war. Deposits in the various counties were described in the 1918 volume of Mineral Resources of the United States.

Georgia

The deposits at Louise, Troup County, have attracted considerable attention of late, but the reserves and grade of the material have not been ascertained.

Montana

A large chromiferous serpentine belt, one-half to three-quarters of a mile wide and 30 miles long, has been reported in Sweet Grass and Stillwater Counties. The reserve of concentrating material[15] has been roughly estimated at 200,000 tons containing less than 40 per cent of Cr_2O_3, but recent concentrating tests apparently failed to produce a marketable product under present specifications. The total reserve[16] for Montana has been estimated at 400,000 to 500,000 tons. Only the west and east ends of the area have been developed, and it is believed by some that the central section warrants further prospecting.

Oregon

Two large serpentine areas containing chromite deposits have been found in Oregon, one in southwestern Oregon in Jackson, Josephine, Douglas, Curry, and Coos Counties, and the other in eastern Oregon along the flanks of the Blue Mountains in Grant, Wheeler, Baker, and Malheur Counties. The reserves in 1925 were estimated at 62,000 tons which included 6,000 tons of ore in stocks. The total production to date has been about 36,500 long tons averaging 40 per cent of Cr_2O_3.

Pennsylvania-Maryland

A belt of chromite-bearing serpentines, extending from south of Baltimore, Md., into southwestern Chester County, Pa., was first exploited by Isaac Tyson in 1827. Operations continued until 1882, the total output being about 250,000 tons of ore containing from 40 to 63 per cent of Cr_2O_3. Nearly 80 per cent of this production came from the Wood Pit mine, in Lancaster County, Pa., and the Reed mine, near Cooptown, Md., the former contributing around 95,000 tons and the latter over 100,000 tons. The remaining output was obtained from a number of smaller mines and sand concentrates made by streams

15/ Diller, J. S., Chromite: U. S. Geol. Survey, Mineral Resources of the United States, 1918, Pt. I, p. 622.
16/ Furness, J. W., Work cited, p. 140.

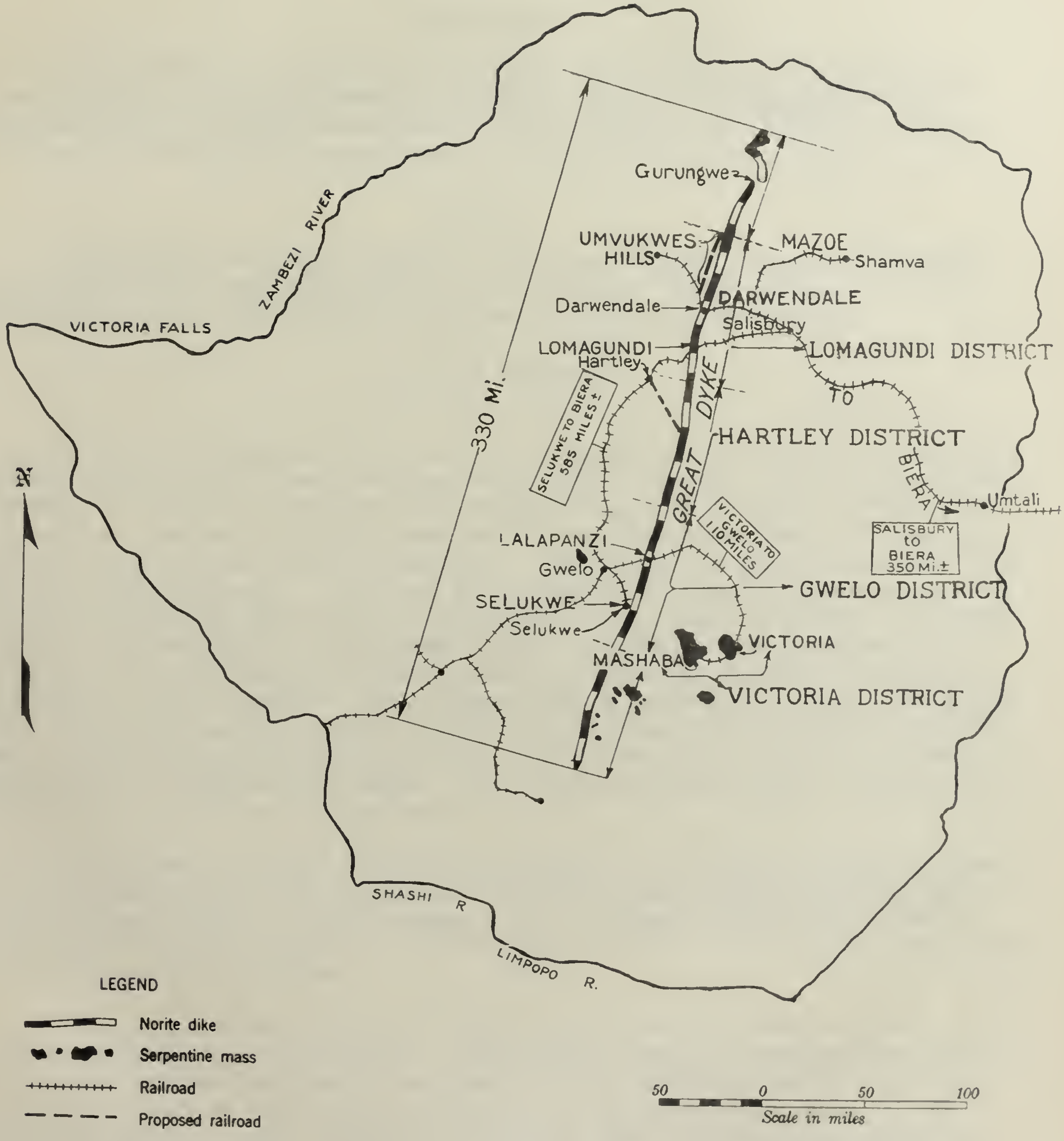

Figure 5.–Chromite deposits of Southern Rhodesia

crossing the chromiferous area. Production from 1881 to 1913 was of minor importance, and only 1,200 tons was produced from 1914 to 1929. The known reserves are small. Geophysical prospecting methods have recently been tried in the field and drilling is now in progress.

Washington

The deposits of chromite in Washington are located on Cypress Island, Skagit County, near Mount Hawkins, Kittitas County, and Okanogan County. The production to date has amounted to 200 tons and the reserve is said to be about 2,500 tons of concentrates running 40 per cent of Cr_2O_3.

Wyoming

Wyoming has one known deposit, in Converse County, which has produced 1,080 tons of ore averaging around 40 per cent of Cr_2O_3. The reserves are estimated at less than 2,500 tons of ore of about the same grade.

Southern Rhodesia

Southern Rhodesia is the world's largest producer of chromite, supplying nearly 44 per cent of the world output in 1929. Production in 1929 amounted to 261,710 long tons, compared with 195,916 long tons in 1928. Of the 1928 production, the Gwelo district produced 79 per cent; the Lomagundi district, 16 per cent; tnd the Victoria district, 5 per cent; the Hartley district produced a small quantity. The total production of Southern Rhodesia from 1906 to 1929, inclusive, was about 1,912,000 long tons. This production has come largely from deposits of high-grade ore which occur in irregular, more or less lens-shaped bodies in the pre-Archean crystalline schists at Selukwe and Mashaba. The deposits of the Great Dyke, a formation of basic and ultrabasic rocks averaging 4 miles in width, and traversing the central part of the country in a north-northeast and south-southwest direction for 330 miles, are made up of regular thin seams. The accompanying map (fig. 5) shows the location of the principal districts.

The Victoria district is the southernmost of the chrome-producing areas. The principal deposits are at Mashaba, 20 miles east of Fort Victoria and several miles east of the Great Dyke. Lack of cheap transportation has handicapped the development of this district, but in 1929 a railway extension from Victoria was completed.

The principal deposit of the Gwelo district, which lies northwest of the Victoria district, is at Selukwe. This deposit has been producing steadily since 1906 and has supplied a large part of the Rhodesian output. The ore occurs in lenses in talc schist, chlorite schist, and serpentines in the older pre-Archean rocks near the Great Dyke. Most of the lenses being exploited range from 150 to 450 feet in length. The following is said to be a typical analysis of Selukwe ore:[17]

17/ Furness, J. W., Chromite: Mineral Resources of the United States, 1927, Bureau of Mines, Pt. I, p. 319.

Cr_2O_3, 51.10 per cent; FeO, 11.40 per cent; Fe_2O_3, 1.40 per cent; MnO, 0.50 per cent; Al_2O_3, 15.20 per cent; MgO, 12.70 per cent; CaO, 0.90 per cent; P_2O_5, 0.50 per cent; S, trace; SiO_2, 4.80 per cent; H_2O, etc. 1.97 per cent.

Another important producer of the Gwelo district is the Lalapanzi area, located about 30 miles northeast of Selukwe. The chromite here occurs in three continuous seams paralleling the Great Dyke. The seams dip at an angle of 25° and are from 2 to 9 inches wide. The grade of the ore ranges from 48 to over 50 per cent of Cr_2O_3. The Gwelo district has adequate rail transportation to the port of Biera, Portuguese East Africa.

The Hartley district which lies about 80 miles north of Lalapanzi, has been a relatively unimportant producer but is believed to contain large reserves. In 1929 three seams were discovered which appeared to be continuations of the Lalapanzi and Umvukwe seams on the south and north, respectively. The seams dip 10° on both sides of the dyke, forming a trough. Two of these seams average 8½ inches in width over a large area. The reserves as developed over a length of about 43 miles and an average inclined depth of 100 feet, were estimated at about 3,000,000 tons averaging 49.3 per cent of Cr_2O_3. The ore is transported 43½ miles to Hartley station by trucks and wagons, but a narrow-gage railroad to improve transportation facilities has been proposed. It was estimated that the cost of delivering ore, c.i.f. European and United States ports, would range between \$18 and \$19.[18]

The Lomagundi-Mazoe district, which lies north of the Hartley area, is the second largest producer of chromite in Rhodesia. Several chromite seams have been found in the Dyke, which extend almost continuously from Darwendale, to the vicinity of the farm Gurungwe, a distance of approximately 70 miles. The continuity of the deposit apparently is broken near Gurungwe, but ore occurs again further north and is said to be plentiful and of high quality. Owing to lack of rail transportation, production in this district has been confined largely to the area immediately north of Darwendale where the railroad crosses the Great Dyke. Recently deposits[19] at Umvukwe, several miles north of Darwendale, have been developed and appear to contain large quantities of high-grade chromite. Early in 1930 the construction of a branch railway from a point near Darwendale northward along the west side of the Dyke was reported. The completion of this railroad is expected to result in a greatly increased production in the Lomagundi district.

Lack of data makes it impossible to appraise numerically the reserves of chromite of Southern Rhodesia, but it can safely be stated that the deposits now developed along the Great Dyke, are capable of producing several million tons of high-grade ore.

18/ South African Mining and Engineering Journal., Important Rhodesian Chrome Development. vol. 40, No. 1996, Dec. 28, 1929, pt. 2, pp. 451-453.

19/ Mining Magazine, London, Umvukwe Chromite Deposits, Rhodesia: March, 1929, p. 181.

The following table gives the production of chromite in Southern Rhodesia from 1906 to 1929, inclusive:

Production of chromite in Southern Rhodesia
from 1906 to 1929, inclusive

Year	Quantity, long tons	Year	Quantity, long tons
1906	3,256	1918	27,934
1907	10,415	1919	31,502
1908	11,927	1920	53,811
1909	22,375	1921	44,810
1910	39,288	1922	83,460
1911	46,753	1923	86,317
1912	61,839	1924	154,219
1913	56,593	1925	121,275
1914	43,042	1926	161,780
1915	54,089	1927	194,658
1916	79,349	1928	195,916
1917	65,145	1929	261,710

Union of South Africa

The production of chromite in the Union of South Africa has increased rapidly during the last few years as a result of the development of large deposits in Transvaal along the margin of the Bushveld Complex at Rustenburg and Lydenburg. Accurate data on the reserves in this area are not available and estimates which have appeared in the literature vary greatly. However, it is probable that there are several million tons of ore in these deposits. The ores for which assays are available are comparatively low-grade, averaging about 43 per cent of Cr_2O_3. Owing to the lower transportation and mining cost, the ore can be marketed at a cost per unit of chromium considerably below the Rhodesian ores. It has not, however, been demonstrated that consumers will remodel metallurgical plants, originally designed to handle 50 per cent ore, in order to take advantage of the cheaper Transvaal chromite. At present the ore finds a market in the chemical and refractory industries and some of the higher-grade ore has been used in the metallurgical industry.

H. Schneiderhohn has described the Bushveld deposits[20] as follows:

> The Bushveld chromites have been known for a good many years, but exploitation of them commenced about six years ago. So far they are mined only in two districts, one being the Lydenburg district,

20/ Mining Journal, London, Chromite of the South African Bushveld: Vol. 169, No. 4940, Apr. 26, 1930, p. 345.

in the eastern part of the Bushveld, and the other the Rustenburg district, in the western part. Most of the chromite bands stretch over a length of hundreds of metres, or even kilometers, without interruption, although faulting is not infrequent, which may cause considerable displacement of the chromite zones. The thickness of the chromite bands varies from a few centimeters to over four metres. Usually there are two or more chromite horizons; in the Lydenburg district there are two, and in places three. In the Rustenburg district there are three chromite bands, one under the other being exploited. ******************

In different localities of the Bushveld numerous prospecting works exploring the chromite zones have been carried out in the last few years. Since 1924 there has been regular exploitation of chromite with a steadily increasing annual production. The productions have been as follows in round figures: 1924, 4,150 tons; 1925, 10,000 tons; 1926, 13,600 tons; 1927, 15,400 tons; 1928, 28,400 tons; and in 1929, about 60,000 tons. It is hoped to bring the annual production up to 100,000 tons in the near future. Last year the Transvaal output was round 16 per cent of the world production of chromite, based on the 1928 world production. The ores are shipped via Lorenzo Marquez, and contain from 40 to 46 per cent Cr_2O_3, averaging about 43 per cent and 28.5 per cent of ferrous oxide. The price of the 42 per cent chromite is about 58s. 6d. per ton c.i.f. in European ports.

The total ore reserves are very great, and are estimated to amount to hundreds of millions of tons.

The chromite is mined, partly by open working and partly from underground, in which case pillars must be left standing to support the roof. In the Lydenburg district the chromite under exploration has a thickness of about 1.20 metres, although in places, as at Grootboom, it is 3.60 metres thick. In the Rustenburg district the chromite has less thickness but has an average Cr_2O_3 value of about 42.5 per cent, which compares with values of 41.5 to 47 per cent in the Lydenburg area. The transport of the chromite to the nearest railway station is rather expensive, but there is said to be prospects of an early extension of railway communication to the chromite mines.

Mention may be made of the fact that a portion of the chromite exploited is from the occurrences of a secondary character formed by the chromite masses which have broken away from the outcrops of primary ore on the high ground and been carried down into the valleys. This is of relatively high grade, containing about 45 per cent of Cr_2O_3.

New Caledonia

Chromite has been mined in New Caledonia[21] since about 1875 and the production up to the end of 1929 has totaled about 1,346,000 long tons, or about 19 per cent of the world total since 1827. Exports in 1929 amounted to 58,212 long tons. There were five[22] operators during 1929, the most important of which was the Societa La Tiebaghi, which operates the Tiebaghi mine at Pagoumene. This mine produces ore carrying 48 to 52 per cent of Cr_2O_3. It is equipped with modern ore-handling devices, but owing to the depth the operating costs are high. Societi Chimique du Chrome operated the Fantoche and Alpha mines in the Pagoumene district. The other operations were those of the Société Caledonia which has mines at Coulée and Plaine des Lacs, that of M. Talou with the Chagrin mine at Koumac, and that of M. Vernier who operated the Alice-Louise mine on the Bay of N'Go. The mines are controlled chiefly by British and American capital. The ore occurs in serpentinized peridotite, and in 1925 reserves were estimated[23] at more than 1,500,000 tons of 50 per cent ore.

New Caledonia is the principal source of the high-grade ore required by the chemical industry.

Cuba

Production of chromite in Cuba began in 1916, and the total output from 1916 to 1929, inclusive, as indicated by imports from Cuba into the United States, was about 213,000 tons, or about 3.0 per cent of the total world output from 1827 to 1929, inclusive. In 1929, Cuban production amounted to about 53,000 long tons, 9 per cent of the world total.

The deposits of Cuba described by Burchard[24] in 1919, are located in Camaguey, Oriente, and Matanzas Provinces. The most important mines lie in the Nuevitas district, in Camaguey, although considerable production has come from several mines around Nipe Bay (Oriente). In addition there are extensive deposits of chromiferous nickel-iron ores in Oriente and Camaguey provinces. The reserves of chromite have been estimated at around 100,000 tons, and there is reported to be 3,000,000,000 tons of chromiferous-nickel iron ore in the Mayari and Moa Bay fields.

The chromite in Cuba is low-grade, averaging about 33 to 43 per cent of Cr_2O_3; 10 to 12 per cent of Fe; 1 to 6 per cent of SiO_2; 20 to 32 per cent of Al_2O_3; and some moisture. Magnesia amounts to 14 to 20 per cent in many ores.

21/ Diller, J. S., Chromite in 1918: U. S. Geol. Survey, Mineral Resources of the United States, 1918, Pt. I, pp. 704-709.
22/ U. S. Department of Commerce, Chrome Mining in New Caledonia: Commerce Repts., No. 25, June 24, 1929, p. 781.
23/ Furness, J. W., Chromite: Mineral Resources of United States, 1925, Bureau of Mines, Pt. I, p. 145.
24/ Burchard, E. F., Chrome-Ore Deposits in Cuba: Am. Inst. of Min. and Met. Eng., Bull. 153, Sept., 1919, pp. 2523-2546.

India

The principal deposits of chromite in India are found in the Quetta-Pishin and Zhob districts in Baluchistan; the Hassan and Mysore districts in Mysore State; and the Singhbhum district in the States of Bihar and Orissa. The deposits in Baluchistan occur in veins and irregular segregations in the serpentine which accompanies the great basic intrusions of Upper Cretaceous age. Much of the ore averages in excess of 50 per cent of Cr_2O_3. In Mysore the ore is found in ultra basic dunite dikes, cutting gneiss, which have been altered in places to serpentine, and generally occurs in veins ranging from 9 to 12 inches in width. The ore usually averages up to 52 per cent. The Hassan ores lie in a talc-serpentine belt about 20 miles long and the ore is mainly in disseminations of low tenor. The Singhbhum deposits often contain up to 53 per cent of Cr_2O_3 and occur in a series of ultrabasic igneous rocks. Other promising deposits have been reported in Bombay Presidency and on the Island of Ceylon. Indian reserves in 1925 were estimated to contain about 800,000 long tons of ore, averaging 50 per cent of Cr_2O_3.

During the last four years, 1926 to 1929, inclusive, production has come from the Zhob, Hassan, Singhbhum, and Mysore deposits; Zhob and Hassan are the most important. The production from 1903 to 1929 has totaled about 616,000 long tons, or 8.7 per cent of the world total since 1827. Production in 1929 amounted to 49,566 long tons which was 8.2 per cent of the world total.

Practically all of the ore produced is exported to Europe and the United States through British Indian ports and the port of Mormugao in Portuguese East India. Production from 1903 to 1929, inclusive, is shown in the following table:

Production of chromite in India, 1903 to 1929
(Long tons)

Year	Quantity	Year	Quantity	Year	Quantity
1903	284	1912	2,890	1921	34,762
1904	3,596	1913	5,676	1922	22,778
1905	2,708	1914	5,882	1923	54,243
1906	4,375	1915	3,776	1924	45,463
1907	18,303	1916	20,156	1925	37,452
1908	4,745	1917	27,061	1926	33,382
1909	9,250	1918	57,770	1927	57,207
1910	1,737	1919	33,439	1928	45,456
1911	3,804	1920	26,801	1929	49,566

Yugoslavia

The production of chromite in Yugoslavia before 1925 was negligible, but during the 5-year period, 1925 to 1929, inclusive, the yearly output

Figure 6.—Chromite deposits of Asia Minor and the Balkan States (Turkey in Asia, Greece, Yugoslavia, Bulgaria, Rumania, Austria, Cyprus, etc.)

averaged 19,015 long tons. All but a small tonnage came from the southern and central Serbian deposits. Two mines in southern Serbia, the Orashje and Gorantsi, furnished nearly the entire Serbian production during this period. The best known deposits are controlled by foreign concerns.

The three principal chromite-bearing areas (fig. 6) in Yugoslavia are:

(1) Central and southern Serbia extending from Uzice, Čačak and Kushevats on the north, to Raskha on the south (lies in the Kapaonik Mountains).

(2) The Serbian Macedonian field, (probably a continuation of the Kapaonik Mountain area) lies mainly about 20 kilometers northwest of Uskub. Other Macedonian mines have been located near Monastir and between Saloniki and Uskub (Skoplje).

(3) Central Bosnia, especially in the Dubostica valley, some 15 to 20 kilometers north of Vares (40 kilometers north of Sarajevo).

Russia

Russia produces sufficient chromite to supply its own needs and a small export trade which amounted to 5,055 metric tons in 1928-1929. Russia consumed nearly 21,000 metric tons of chromite in 1926-1927, and the apparent consumption (production minus exports) was 26,555 tons in 1927-1928, and approximately 23,000 tons in 1928-1929. The following table shows the production of chromite in Russia in 1913 and from 1922 to 1928, inclusive:

Production of Chromite in Russia, by districts, 1913 and 1922 to 1928

(Metric tons)

Regions	1913	1922-23	1923-24	1924-25	1925-26	1926-27	1927-28
Ural Province	24,920	875	8,773	22,117	26,622	13,015	20,782
Gov. of Orenburg	-	-	-	87	-	-	8,401
Bashkirian Aut.S.S.R.	1,296	17	3,121	4,991	3,740	6,279	353
Kazaksky Aut.S.S.R.	-	-	-	2,916	-	-	-
Total	26,616	892	11,894	30,111	30,362	19,294	29,536

Comprehensive data on the chromite resources of Russia are not available, but owing to the large areas of basic rocks in which chromite is often found, it is probable that the resources of Russia will prove to be large. In recent years the Soviet Government has been very active in prospecting for and developing new deposits.

Deposits of chromite are known in the Ural area, the Northern Caucasus, the hinter Caucasus, and the hinter Baikal. The Ural deposits, which are the most important at present, are found in serpentines and soapstones, and extend from Perm southward into Orenburg. The reserves of this area have been estimated at 6,000,000 tons.25/

It has been estimated that the largest of the Ural deposits, Saranovsky, near Perm, contains a total of 1,457,000 tons of developed, probable, and possible ore ranging between 40 and 50 per cent of Cr_2O_3. In addition there is a considerable tonnage of lower-grade concentrating ore. Second in importance is the Gologorsky deposit, near Sverdloosk. A concentrating plant was built at this property in 1914, but operations were suspended shortly after and were not resumed until 1926. The concentrates contain from 42 to 48 per cent of Cr_2O_3, and are utilized mainly by the Shaitansky bichromate manufacturing works.

According to V. Kozlov, the Beloretzky Mining Trust in 1927 was mining a deposit of high-quality ore on the eastern slope of the Ak-bik Mountain, in the Bashkirian Autonomous Province. The mines are located 5 to 6 kilometers southwest of the village of Shigaev and 30 kilometers from the Beloretzky plant where the ore is being used in the manufacture of bichromate.

Numerous deposits have been discovered recently in the Chaliloosky region which extends into the Orenberg territory. These deposits are of higher grade than other Ural deposits and in many cases the ore can be shipped directly.

Other Ural deposits are located in the Novoe Delo, Revdinsky, Zlatoustovsky, Ubalinsky, Miassky, and Troitsky districts. The grade of the ore in these deposits ranges between 20 and 41 per cent of Cr_2O_3.

Many chromite deposits have been found in northern and hinter Caucasus, but as yet there has been little production. Recently chromite associated with tin ore was found on Djoumarakli-Tuba Mountain in the Arhizky region, northern Caucasus. Many deposits have been found in Siberia, especially in the Baikal area, but these also have been unproductive to date.

Greece

The chromite ores of Greece are of considerable importance; the principal deposits are in Thessaly and in the Khalkidike Peninsula. In some years as much as 15,000 tons of ore has been mined. The ore occurs in irregular masses in serpentines and other ultra basic rocks. The irregular nature of these deposits has made exploitation sporadic. The principal mine in Thessaly is close to Pharsala, 30 miles west of Volo, the port from which Thessalian ore is shipped to Belgium and the United States; within recent years this mine has produced annually more than 10,000 tons of 38 to 40 per cent ore.

25/ Trinkler, G. V., Chromite: Annual Report on the Mineral Resources of the U.S.S.R., 1926-1927, pp. 1054-1060.

Another important mine in Thessaly is the Alchani Domokos; in 1927 it produced considerably more than the Pharsala mine. On the south side of Lake Daoukli at Nezero is a smaller mine close to the main railroad line from Larissa to Lamia; numerous smaller deposits occur in the Neochori district on the western slope of the Othyris Mountains, at Voivoda, near the narrow-gage railroad from Kalabaka to Volo. A number of comparatively small workings are found in Macedonia, 20 miles west of Salonika, near Katerini, where the ore occurs as segregations in serpentine derived from peridotites. Ore bodies of similar nature occur in places on the Khalkidike Peninsula. The mine at Vavdos near Glatista has an average annual output of 300 tons. Deposits have been reported in several parts of Euboea and Boeotia and from the Island of Skyros.

The following table shows the production of chromite in Greece by various periods, 1881 to 1928, inclusive, in long tons:

Production of chromite in Greece, 1881-1928,
(Long tons)

Year	Total	Annual average
1881-1890	1/ 3,582	358
1891-1900	20,721	2,072
1901-1910	94,226	9,423
1911-1920	76,599	7,660
1921-1925	54,063	10,813
1926	19,732	19,732
1927	17,041	17,041
1928	20,622	20,622
1929	23,832	23,832

1/ Estimated.

Turkey

Turkey has been an important producer of chromite since the middle of the nineteenth century. Its output led the world from 1860 until the development of the New Caledonian deposits during the letter part of the century. Since then Turkish production has been dwarfed further by India and Southern Rhodesia. The estimated total output from Turkish mines is about 832,000 tons, or 11.7 per cent of the world total from 1827 to 1929, inclusive. Annual production is expected to increase materially as soon as the proposed railway system is completed. The reserves of chromite are not actually known. Estimates range from 1,000,000 to 15,000,000 tons of ore averaging 50 per cent of Cr_2O_3.

Figure 6 shows the chromite deposits of Turkey. The principal mines are in the northwestern part of the country near Brusa. Several mines have been developed which contain ore running 50 to 52 per cent of Cr_2O_3. Other

high-grade deposits are found near Adalia and Makri. Recently there has been much interest in the deposits east of Eskishehir where several mines containing 43 to 50 per cent ore have been found. The chrome industry of Turkey was described in 1929 by Von Engelmann.[26]

The following table shows the production of chromite in Turkey, from 1881 to 1925, by 5-year periods, and for 1926, 1927, 1928, and 1929.

Production of chromite in Turkey, from 1881 to 1925, by 5-year periods, and for 1926, 1927, 1928 and 1929 [1]
(Long tons)

Period	Production	Annual average	Period	Production	Annual average
1881-1885	47,542	11,886	1916-1920	71,008	14,201
1886-1890	97,722	19,544	1921-1925	23,075	5,769
1891-1895	117,812	23,562	1926	6,565	6,565
1896-1900	56,075	11,215	1927	18,029	18,029
1901-1905	53,148	10,630	1928	11,662	11,662
1906-1910	109,248	21,850	1929	14,840	14,840
1911-1915	105,774	21,155			

1/ Figures subject to revision.

Japan

The principal chromite deposits in Japan are the Wakamatsu mine in Hoki Province; Niimi and Takase mines in Bitchu Province; and the Lasaguri and Mundagao mines in the Province of Chikuzen. The ores occur as detrital deposits and as segregations in serpentines cutting gabbros and peridotites. The ore is said to average 40 per cent of Cr_2O_3.

Production began in 1907, and the entire output has been consumed in Japan. The total production to the end of 1929 was nearly 112,000 long tons, 1.6 per cent of the world total from 1827 to 1929, inclusive. Production during 1929 amounted to 9,018 tons.

Summary

The world reserves of chromite are not susceptible to precise estimation, but the supply of ores of marketable grade which exists in Southern Rhodesia, Turkey, India, Union of South Africa, Russia, and New Caledonia is believed to be sufficient to meet the world's needs for several decades. Reserves of minor importance exist in Cuba, Yugoslavia, Greece, and Japan. Some countries contain deposits which are too low-grade to be of commercial importance under present market conditions. In addition to these supplies a large quantity, possibly as much as 5 billion tons, of chromiferous iron ore of the Mayari type exists in Cuba, Porto Rico, Celebes, Borneo, Philippine Islands, Gold

26/ Von Engelmann, H., Turkey Extending Railroads to Develop Chrome Resources: Eng. and Min. Jour., vol. 127, No. 26, June 29, 1929, pp. 1037-1038.

Coast, Greece, and Australia. While their chromium content is of negligible economic importance at present, these ores will probably constitute an important source of chromium when the high-grade deposits are exhausted.

POLITICAL AND COMMERCIAL CONTROL

Political Control

Of the 1929 world production of chromite about 63 per cent originated in the British Empire, 10 per cent in French territory, 9 per cent in Cuba, 7 per cent in Yugoslavia, 5 per cent in Russia, and the remaining 6 per cent largely in Greece, Turkey and Japan. The British Empire's proportion of the world's high-grade chromite reserve is probably larger than its share of world production in 1929.

Commercial Control

Africa. - The large chromite reserves of Southern Rhodesia and the Union of South Africa are owned mainly by the Chrome Corporation (Ltd), which is controlled by the Edmond Davis group (British). Less important areas are being developed by the Bee Chrome Corporation (Ltd.), Mann-Little & Co., and the Becker Trust Co. (all British). American interests are reported to own considerable stock in some of these companies.

Oceania. - In New Caledonia, the Teibaghi Mine at Pagoumene, is operated by the Société La Tiebaghi (Edmond Davis group (British)), and the Fantouche and Alpha mines in the same district are being exploited by the Société Chimique du Chrome (American). There are also a number of small French and native operators.

Asia. - In Turkey, where reserves are reported to be large, brisk competition has developed for concessions. The Roechling group (German) has acquired large holdings in the Dagh Ardy sector of the Brusa field. The eastern half of this field is jointly controlled by the Edmond Davis group (English), a syndicate of Swedish iron companies, and the German Krupp company. A French group has become interested in the Eskishehir district, while mines further north, in the Bolsular district, have been conceded to a Turkish company (recent reports indicate that the Roeschling interests have acquired the holdings of the Turkish concern). The Makri deposits are operated by a French company, Société Miniere de Fethie, while the Smyrna mines have been acquired by the Patterson Brothers (British). An American company is reported to have obtained a concession in the Brusa district.

The Japanese chromite deposits are controlled and operated by Japanese interests, and the Indian deposits are largely controlled by British capital.

Europe. - Production in Yugoslavia is controlled largely by German and French interests, while that in Greece is dominated by French, American, and Grecian capital.

North and South America. - The deposits of chromite in North and South America are controlled largely by American interests. Cuba is the only important producer.

I. C. 6567 March, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MEDICAL SERVICE, ACCIDENT REPORTS, COMPENSATION, AND WELFARE AT IRON MINES IN THE LAKE SUPERIOR REGION

BY

F. S Crawford

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I.C.6567,
March, 1932

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MEDICAL SERVICE, ACCIDENT REPORTS, COMPENSATION, AND WELFARE AT IRON MINES IN THE LAKE SUPERIOR REGION[1]

By F. S. Crawford[2]

The wide variation in the medical facilities, type of accident reports and provision for the welfare of employees, in addition to the compensation provided by the various State laws, was studied by the writer in the Lake Superior region.

Several companies furnished the information which follows. Names have been omitted, but the information should be of interest to companies desirous of improving this branch of their accident-prevention work.

Because a summary of the methods used would merely result in an attempt to set up an ideal, it was concluded that in preparing this paper a statement of the individual methods of each company would be of the most interest, and they have been listed with no regard to the size of the company.

In the appendix are a number of report forms used by various operators for reporting accidents to the company and to the State. These are self-explanatory and many are mentioned in the body of circular.

COMPANY A

Medical Service:- All new employees of company A are required to take a physical examination, and all old employees must be reexamined once every two years. Dispensary service is available at the mine location with a full-time doctor in charge. All hospital cases are taken to the general hospital, which is 6 miles from the property.

Slight injuries are treated in the mine by first-aid men who see that the injured man is taken to the doctor for further treatment if necessary. First-aid supplies are kept under ground on every level. Men are required to report all injuries, regardless of triviality, on the timekeeper's check sheet by marking whether they have been injured during the day.

Accident Reports:- Reports of injuries are made by the foreman and by the doctor, and a report is made to the State. Copies of these reports are attached. There is no workmen's committee report of an injury. The classification of accidents used by the State is used by company A. In classifying lost time, the day of injury is not counted as lost time, as a man is paid full time for the day on which he is injured.

Return-to-Work Policy:- When a man returning to work is not able to take up his regular job, he is given employment which he can do with the same pay he received at the time of his injury.

Reports on severity are kept by the safety department; a copy of such a report is included in the appendix (30) to this report.

Compensation and Welfare:- Compensation is governed by the State compensation law. Company A carries its own insurance. An accident fund and a death fund, according to

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6567."

2 - District engineer, United States Bureau of Mines Safety Station, Duluth, Minn.

State compensation requirements, is set up, based upon a certain percentage of the payroll, according to the previous accident experience of the mine.

COMPANY B

Medical Service.- All men are examined by company B physician when they are hired. Hospital and dispensary service are maintained. In case of accident, the doctor is called to the scene of the accident as soon as possible.

Compensation.- The company is a self-insurer under the State workmen's compensation law and has no funds or benefits other than those compensable by State law.

This company has had no fatal accidents for 5 years.

COMPANY C

Medical Service.- There is no physical examination of either new or old employees of company C. Hospital and dispensary service are available in a town close by; in treating slight injuries, first aid is given in the mine by the shift boss. The man is then sent to the hospital.

The doctor makes a report of the injury if a man is sent to the hospital. The man is paid full time for any part of one-half shift on the day injured. In other words, if he was injured in the morning, he would get a half-shift's pay, and if injured after lunch he would get a full-shift's pay. Generally, the man returns to his regular job, but sometimes it is possible for him to return to light work until he is able to resume his regular occupation. The doctor makes reports on severity to estimate further disability.

In hiring new men in such a small organization, safety instructions are necessarily given by the foreman, who also uses his judgment as to placing men on hazardous jobs.

Compensation and Welfare.- Company C is insured by a regular insurance company. No pension funds, death funds, or accident funds are maintained through the cooperation of the company.

COMPANY D

Medical Service.- Company D requires that all employees be examined physically at least once a year. Hospital service is available at convenient places at each of the mines operated by the company. Treatment for slight injuries is taken care of through first-aid equipment at various points about the mines. The company insists that every accident, no matter how slight, be reported to the management; the case is then followed up and investigated by the company inspector to see that medical service is given where necessary.

Accident Reports.- The first report of an accident is made by the foreman in charge. If the accident requires the doctor's attention, he also makes a report on it. The safety inspector investigates the cause of the accident and reports to the management. The whole matter is then given general review in conference at the foremen's meetings.

If the man loses more than the shift upon which he is injured, the accident is classed as a lost-time accident. Men are paid for the day on which they are injured. When a man is ready to return to work, arrangements are made to place him temporarily until he is able to resume his regular work, provided that such action is thought advisable. The doctor's reports are received monthly on serious cases with estimates as to further disability causing lost time.

Compensation and Welfare.- Company D carries its own compensation insurance under the State workmen's compensation act. In addition, there is maintained for the benefit of the employees a pension fund and a benefit association. This applies to employees in case of sickness, or accidents occurring outside of employment.

COMPANY E

Medical Service.- Company E gives new employees a physical examination at the time they are employed, and each year thereafter. Hospital and dispensary service are available at a hospital within 5 miles; emergency cases are taken to a doctor in a nearby town.

Accident Reports.- Reports of injuries are made by the doctor and by the safety engineer at the time of the injury, upon forms supplied by the insurance company. In classifying the accident as to lost time, company E states that the day upon which the man is injured is classed as lost time and that he is not paid full time for that day. In returning the man to work he is given a job at light work which he can handle until he is able to resume his regular occupation. The doctor makes a report on the severity of the accident in the form of an estimate at the time of the injury and a final report when the man returns to work.

Compensation and Welfare.- Company E has no pension fund, but carries its insurance with an insurance company. The employees are insured in addition under a group policy by a life assurance society; this policy is a life and health and accident policy.

COMPANY F

Medical Service.- Company F requires a physical examination of each new employee and annual examinations of old employees. Hospital and dispensary service are provided through contract with a local clinic. All injuries, however slight, the company states, call for medical attention.

Accident Reports.- The doctors make a preliminary and a final report on all injuries; the foreman describes every accident on his daily report. There is no investigation of an accident or report by a committee of workmen.

In classifying the accident, the day of injury is not classed as lost time, as the man is paid full time for the day injured. In returning men to work they are given a job at light work until they are able to resume their regular jobs. The doctor makes a report on the severity of the accident in his preliminary report to estimate the time the man will be away from work.

Compensation and Welfare.- The workmen's compensation law of the State governs all compensation payments. In addition to compensation which company F carries with the insurance company, the company also has arranged for group insurance of its employees, with the following features:

> Employees are eligible for group insurance after three months of employment, when subscription to this insurance is required of all employees. No medical examination for this insurance is required. Benefits are $12.50 per week for sickness and accident at a cost of $1.04 per month. Death benefits are from $1000 to $4000, according to the salary earned by the employee, at a cost of 60 cents per month to the employee and 43 cents per month to the company. If an employee becomes totally and permanently disabled before reaching the age of 60, the policy becomes due and payable.

COMPANY G

Medical Service.- Company G requires physical examination of new men, but does not examine the men periodically thereafter. Hospital and dispensary service are available in a nearby mining town. When a man is injured he is sent to a doctor in the town. First-aid material is available underground near both shafts and some is available in the change house.

Compensation and Welfare.- Company G is a self-insurer under the State compensation law, but in addition carries group life insurance for its employees under the yearly renewable term plan with an insurance company.

COMPANY H

Medical Service.- Physical examinations of new employees are given upon employment by company H, and each employee is given a reexamination every year. Hospital and dispensary service are available to the men. All slight injuries are given first-aid treatment, and the injured then sent to the doctor.

Accident Reports.- Reports of the injury are made by the doctor, by the foreman, and by the safety department. There is no workmen's committee report on an injury. A man is paid full time for the day he is injured, the day of injury not being counted as lost time.

If a man is willing and able to take a light job before he is able to return to his regular job, he is returned to work at the light job; this is entirely the workman's choice, however, and works to his own benefit. Naturally, the man working at the light job earns more than the compensation rate. Thus, the injury is not classed as lost time as the man is returned to work, regardless of whether he has or has not been returned to his regular job.

The doctor makes a report on the severity of the accident in the way of an estimate of the time the man will be away from work. In case of doubt the workman is given the benefit thereof.

Company H tries, as far as possible, to give men work which they are mentally and physically capable of doing, avoiding the placement of men with serious defects in hazardous positions.

COMPANY I

Medical Service.- A system of physical examinations for both new and old employees has been established by company I. Hospital and dispensary service are available to the men. For slight injuries, first aid is given at the mine, and if the injury is considered severe, the man is then sent to the doctor.

Accident Reports.- The first report of an accident is made by the foreman. The doctor also makes a report of the injury when the man comes to him for treatment. No report is made by the safety department nor is any report of the injury made by a committee of workmen.

The men are paid full time for the day they are injured, so that the day of injury is not classed as lost time. If a man fails to return to work the day after the accident, it must be classed as a lost-time injury.

In returning a man to work he is given any job where he can do a day's work without being further endangered by his injury. However, it is not the policy of company I to create a job to take care of an injured man. The fact that a man returns to lighter work than his regular job has no effect upon the classification of the accident as a lost-time accident. Regardless of the amount a man earns if he is on light work, it is classed as a lost-time accident if he does not return to his regular work on the day following the day of injury.

Compensation and Welfare.- The mines operated by the company are in Michigan, Wisconsin, and Minnesota and are guided by the compensation laws of these States.

Company I does not maintain or operate a pension fund; it is self-insuring. In addition to the compensation which the law requires, all employees carry group insurance through the mining company, the minimum being $1000 which is paid in full at death. The insurance rate varies with each property and with the accident rate which is based upon the previous 5 years experience at each property.

COMPANY J

Medical Reports.- Physical examination of all employees was begun by company J in February, 1929, and has been continued. All new employees are examined by company physicians

before going to work. In the examinations of the old employees no one was discharged following the discovery of weaknesses, but a fair number whose physical condition was not up to standard were changed over to other work which they could do with safety. Physical examinations are held confidential and are accessible only to the general manager, the head of the pension department, and the district superintendent.

There are hospitals at all company J mines, most of them being operated by the physicians themselves. At the headquarters of the company, a new hospital was built in 1918; this is operated by the company and serves the district. The physicians are paid a salary, and the operation of the hospital is entirely by the company.

The doctors keep their offices in these various hospitals and all patients who are able to do so come to the hospital for consultation or treatment; in other cases the physicians go to the homes. The men pay $1.25 per month for medical services for themselves and the immediate members of their family. This provides for medical attention and necessary medicines. The only extra charges to the families are for cases of confinement, which are $15, and for sickness or injury which are the result of intoxication, and treatment for venereal diseases; these are paid for in addition. In cases of elective operations, anyone going to the hospital pays a regular established rate and a small operating fee. This charge is much less to the employees of the company than it is to outside people who may be admitted to the hospital. Outside patients are treated as private patients of the physicians on the company staff.

In addition to the $1.25 paid by each man per month, company J pays 50 cents per man per month to the physicians for the care of mine injury cases. This is considerably in excess of the actual cost of this service at the present time, but it has been established at this rate to cover any possible contingencies, also to assist in providing reimbursement for the physicians.

There are a large number of men trained in first aid at all of company J properties. A new group of men is continually being trained, so that at present there is a large number of first-aid trained men at every mine. In case of an injury, the man gets first-aid treatment and if it is at all serious he is directed to report at once to the physician. The emergency work is done by the first-aid man, and the injured person is sent at once to the hospital.

Accident Reports.- Each doctor makes a report of every man who comes to him with an injury. In Michigan a weekly report is required from the doctor upon each case, as it is necessary to pay compensation weekly. In Minnesota a report is required but once a month, as the compensation in that State is paid monthly. An original report on each accident is made out by the foremen on a form headed "Report of Personal Injuries to Employees," a copy (5) of which is attached. A copy of the form used for the physicians' reports (13) of the accident is also attached.

In all cases the doctor makes a full record of the injury with the estimated time of disability. The company has gone over its accident records which have accumulated since it came under the compensation law of Michigan in September, 1912, and the compensation law of Minnesota in 1913. It has tabulated the duration of disability from various specific accidents, such as a broken leg and a broken arm. In making comparative statements of accident severity the figures derived from these records have been used, considering at the same time the estimate of the doctor in the particular case.

Reports of slight accidents which do not involve over one day's lost time are made upon a separate form. If a man is off over one full day on account of an accident, the report is made out upon the long report form. A separate report upon each serious accident is made by a mine safety committee, consisting of three men at each mine, on a separate form. The committee, besides investigating the cause of the accident, makes recommendations to prevent a

recurrence and submits a report to the superintendent who reports his action to the manager and the safety department.

When accidents occur to equipment a report is made upon it, regardless of whether or not an injury was sustained. This report is made on the attached form, "Report of Accident to Equipment" (6).

In classifying the accidents the day of injury is not counted as lost time, the men receiving full compensation for the remainder of the day.

Return-to-Work Policy.- An effort is made to get an injured man back to some work as soon as possible following the injury, because the company has found from experience that a man is better off when engaged in some useful occupation than he is when idle; provided that the work upon which he is placed does not aggravate the injury, this company has found that being occupied at some work which an injured man can do hastens his recovery. This return to work is arranged by the agent of the compensation department and the officials of the mine in connection with the mine physician.

It frequently happens that a man is temporarily put on some other work than his regular job, or some lighter work, until he has enough strength and ability to enable him to do his regular work. The Michigan law provides that if a man returns to work at a different occupation than the one at which he was injured and earns less wages than he was earning at the time of the injury, the company is obliged to pay him weekly two-thirds of the difference between the rate he was earning at the time of injury and the rate at which he is temporarily reemployed.

The company has operated a pension fund since January, 1909. There is a pension board which passes upon all applications for pensions, the requirements of which are 25 years or more of service and 65 years of age. The monthly pension is 1 per cent for each year of service, based on the average monthly wage for the last 10 years.

Company J has no provision for death payments outside of what is required by the compensation laws. However, frequently it helps families in case of emergency.

For many years an accident fund was maintained at the properties of the company. The men paid into this fund 30 to 50 cents per month and the company contributed an equal amount. Payments were made from this fund in cases of accidents which occurred to men while at work for the company. When the State compensation laws were enacted, the collection of this fund was discontinued, the amounts on hand being paid out as originally planned for in cases of accidental injuries. During this period the compensation payments required by law were much lower than they are at present, so that the men who had contributed to these funds were benefitted to that extent.

Insurance was carried at the Minnesota properties of company J before enforcement of the compensation law. Since coming under the compensation law, however, under the accepted provisions of the law, the company has carried its own risk and paid the compensation direct to the man or his dependents.

COMPANY K

Medical Service.- New employees are given a physical examination before employment by company K. If the men leave its employ and return after one month's absence they are required to take a reexamination. Old employees are reexamined at stated periods. Hospitals and dispensary service are available to the men at all mines. Men are given first aid at the mine in case of a slight injury and then sent to the doctor for further treatment. Failure to report any accident, no matter how trivial, is considered an infraction of the company rules and sufficient cause for dismissal.

Accident Reports.- Reports of accidents are made by the doctor, the safety department, the foremen, and the workmen's committee. A report is made to the State industrial commission. In case an injury requires a man to quit his work for the day, he is paid full time for that day, and lost time does not start until the following day.

When a man is injured he is not allowed to return to work until the doctor reports that he is fully able to return to the job he left. It is not the policy of the company to place injured men on light work or to transport them to the job if they are unable to travel. Rarely is a man given work at some other job before he can return to his regular job and if it is done, it is not done to conceal or lessen the length of disability. Company K states that it never puts a man on light work for the purpose of obtaining a better accident disability record.

At the time the injured man reports to the doctor.he estimates the number of days that the man will be away from work and states this in his accident report. After 15 days the doctor makes another certificate checking this estimate. If the injured man is able to return to work before 15 days, a final certificate is given by the doctor stating the time the injured man went back to work.

Compensation and Welfare.- Compensation is based upon the compensation laws of Minnesota and Michigan. The company has not established any separate pension fund, death fund, or accident fund, basing its payments entirely upon the provisions of the State compensation laws.

Company K is a self-insurer and rates its own mines and the compensation and insurance fund upon the previous experience of the mine, setting aside enough money to take care of probable accidents.

COMPANY L

Medical Service.- All new employees of company L are given a physical examination before they are hired; all old employees are given an examination by the mine doctor once a year. Any man who quits his job, or is discharged, or any man who takes leave of absence for over 30 days must be reexamined as a new employee. This does not apply to men who are laid off due to a temporary shutdown.

The mine doctor has until recently had a hospital of his own where the mine patients are taken care of. A new 32-bed hospital was built by the mining companies of the district in 1930.

All cuts, scratches, bruises, and minor injuries are treated by first-aid men or by the safety engineer in the dry house. If the injury is of such a nature that it requires the doctor's attention, the patient is sent at once to him for treatment. The doctor decides whether a man returns to work after an injury. An accident check-up system has been perfected whereby it is known at the end of each shift whether an accident of any kind has occurred. This system originated as a result of serious infections developing from small cuts and bruises of which the foreman had no knowledge. With this system in force, no injured employee can leave the property without first receiving medical treatment. A slip is put up in the dry house each day with each employee's check number printed upon it. If a man receives a cut, bruise, or any small injury, he puts an "O" after his number as he leaves at night; if he received no injuries whatsoever, he places an "X" after his number. Whether he is injured or not he must mark the card.

Accident Reports.- The doctor makes a report of the injury as to the nature and the cause. The safety department gets a report of the injury from the underground foreman, and the foreman's report is also sent to the office.

In classifying accidents, the day of injury is not counted as lost time if the man is paid full time for the day injured.

In returning a man to work, he often returns to a job that he can do until he can resume his regular job. As long as the man is actually employed by the company, whether at the same job at which he was injured or at some other job, it is not classified as lost time. The severity of the accident is determined by the doctor, and it is he who decides whether the man is fit for shop work, underground work, or if he is able to go back to work at all for some time. He must present a clearance from the doctor giving him permission to return to work before he is allowed to work by the foreman. Severity and frequency charts are made up each month.

Compensation and Welfare.- Compensation is covered by the provisions of the State compensation law which allows compensation after absence of 7 days due to injury, the amount varying with the nature of the injury. Company L has no special pension fund or death fund. It is self-insuring under the State compensation law and sets up a reserve to compensate for accidents by charging a certain percentage of the payroll each month.

Inasmuch as the company is self-insuring, any reduction in the accident rate has a direct effect upon the insurance rate. If the percentage of the total payroll is not large enough to set up a reserve, it is increased.

COMPANY M

Medical Service.- Company M requires every new employee to pass a physical examination and also reexamines all old employees every year. If a man is laid off over 30 days he is reexamined upon reemployment. Hospital and dispensary service are available in a nearby city. This service is furnished on contract by the doctor in charge. The doctor furnishes medicine as part of his contract; the company furnishes X-ray equipment and power for operation, maintains an ambulance and pays special doctors' fees in addition for certain operations.

For slight injuries every foreman and trammer boss is supplied with a pocket first-aid kit; larger supplies are kept at convenient points underground; this includes splint boxes, first-aid boxes, and blankets. First-aid treatment by the foreman is always followed by treatment at the dispensary.

Accident Reports.- Several forms for reports on accidents and diseases, used by company M are attached in the appendix. A report is made by the foreman on separate forms (8, 9, 10). The safety department makes a first report (7) to the industrial commission; this is followed by a supplementary report (19) and a final report (27). The doctor also makes a final report (20) to be sent to the industrial commission when the disability exceeds three weeks or where there is permanent disability.

There is no investigation of the accident by a workmen's committee. The company gets a receipt from the injured man upon the forms attached, showing that compensation has been paid or that final settlement has been made (26, 28, 29).

In classifying accidents, the day of injury is counted as full time for the man, regardless of what time of the day he was injured. In returning men to work, company M has formed a policy of getting men back on the job as soon as possible, even though they can do only light work. It has been found that men are rehabilitated more quickly by this policy. In addition to getting the man back to his regular work more quickly, this plan also gives the man an opportunity to make more wages than he would receive in compensation alone.

Estimates of the severity of cases are made by the doctor in the less severe cases.

Compensation and Welfare.- Compensation is governed by the provisions of the State workmen's compensation law.

The attached form (31) will be of interest showing how a record of compensation payments is kept in the company office. Another form (30) is attached showing the record of frequency and severity of accidents. The company does not have any separate pension funds, or

death funds, or accident funds. It is, however, self-insuring under the provisions of the State compensation law and maintains a fund able to take care of compensation payments according to the experience of the company in accidents.

PROVISIONS OF MICHIGAN AND MINNESOTA COMPENSATION LAWS

Michigan

Compensation and Welfare.- The general provisions of the State compensation law in Michigan are that compensation shall be 66-2/3 per cent of the wages paid at the time of the injury, with a maximum compensation of $18 per week. No compensation is paid for the first week of the injury, unless the disability causes a loss of time of 6 weeks or more; in this case the first week is then paid for. In case of death payment is made at the regular rate for 300 weeks, if there are any dependents. If there are no dependents to receive payment in case of a fatality there is no payment made except a payment of $200 toward the funeral expenses, which is payable in all cases. In case of total disability the compensation is continued at the regular rate for 500 weeks. The law requires that the company provide medical aid for injuries which last up to 90 days. Many companies however, continue the medical service even if it extends beyond the 90 days. Occupational diseases are not covered by the Michigan law.

Minnesota

In Minnesota, compensation is paid at the rate of 66-2/3 per cent of the wages with a maximum compensation rate of $20 per week. The waiting period in this State is one week, as it is in Michigan, but payment for this is added if the injury extends beyond four weeks. In case of death the compensation payments are continued for 300 weeks if there are dependents. If there are no dependents, a payment of $200 is made to the State treasury, this money being used for payment of disability cases not covered by the law. In addition, a maximum of $150 is required to be paid toward burial expenses. In death cases, where compensation is paid, it is paid monthly until $7,500 has been paid, regardless of the rate per week. There is a specified rate for families of different sizes, all subject to the same maximum of $20 per week. If a widow re-marries and leaves no other dependents, a lump sum is paid to her of one-half the compensation remaining due, but not to exceed two years' compensation. If there are dependent children, the compensation continues to them until the end of the period.

There are many provisions in both the Michigan and Minnesota laws, such as a definite maximum of accident payments for specified injuries, as the loss of a hand, foot, arm, leg, or any other member. For details of these amounts reference should be made to the compensation laws of these two States.

APPENDIX

The following figures are reproductions of various accident forms used in the Lake Superior region:

1. Timekeeper's check sheet.
2. Doctor's first report of accident.
3. Doctor's first report of accident.
4. Surgeon's report of accident.
5. Report of personal injuries to employees.
6. Report of accident to equipment.
7. Mine safety committee's accident report.
8. Foreman's report of accident.
9. Foreman's report of accident and first-aid accident report.
10. Foreman's accident report and company accident report.
11. Record of accidents.
12. Report of slight accident.
13. Physician's report of accident.
14. Final certificate for payment for injuries.
15. Physical examination card for old employees.
16. Employer's first report of injury.
17. Application for employment, physical examination card, and physical examination card for new employees.
18. Industrial commission; first report of accident or industrial disease.
19. Supplementary report on accidents and industrial diseases.
20. Physician's report on accident or industrial disease.
21. Supplemental report of fatal accident.
22. Agreement to pay compensation to dependents on account of death of injured employee.
23. Agreement with injured employee in regard to compensation.
24. Report of compensable accident.
25. Report of noncompensable accident.
26. Receipt on account of compensation.
27. Final report of accident.
28. Receipt on account of compensation for injury.
29. Settlement receipt.
30. Frequency and severity table.
31. Summary of compensation data.
32. Monthly record of accidents.
33. Record of payments as per award of industrial commission for death or disability.

1000, 2-30 GPC

THE MINING COMPANY

Timekeeper's Check Sheet

______________ Mine Date ______________ 193__

1	51	101	151	201	251	301	351	401	451
2	52	102	152	202	252	302	352	402	452
3	53	103	153	203	253	303	353	403	453
4	54	104	154	204	254	304	354	404	454
5	55	105	155	205	255	305	355	405	455
6	56	106	156	206	256	306	356	406	456
7	57	107	157	207	257	307	357	407	457
8	58	108	158	208	258	308	358	408	458
9	59	109	159	209	259	309	359	409	459
10	**60**	**110**	**160**	**210**	**260**	**310**	**360**	**410**	**460**
11	61	111	161	211	261	311	361	411	461
12	62	112	162	212	262	312	362	412	462
13	63	113	163	213	263	313	363	413	463
14	64	114	164	214	264	314	364	414	464
15	65	115	165	215	265	315	365	415	465
16	66	116	166	216	266	316	366	416	466
17	67	117	167	217	267	317	367	417	467
18	68	118	168	218	268	318	368	418	468
19	69	119	169	219	269	319	369	419	469
20	**70**	**120**	**170**	**220**	**270**	**320**	**370**	**420**	**470**
21	71	121	171	221	271	321	371	421	471
22	72	122	172	222	272	322	372	422	472
23	73	123	173	223	273	323	373	423	473
24	74	124	174	224	274	324	374	424	474
25	75	125	175	225	275	325	375	425	475
26	76	126	176	226	276	326	376	426	476
27	77	127	177	227	277	327	377	427	477
28	78	128	178	228	278	328	378	428	478
29	79	129	179	229	279	329	379	429	479
30	**80**	**130**	**180**	**230**	**280**	**330**	**380**	**430**	**480**
31	81	131	181	231	281	331	381	431	481
32	82	132	182	232	282	332	382	432	482
33	83	133	183	233	283	333	383	433	483
34	84	134	184	234	284	334	384	434	484
35	85	135	185	235	285	335	385	435	485
36	86	136	186	236	286	336	386	436	486
37	87	137	187	237	287	337	387	437	487
38	88	138	188	238	288	338	388	438	488
39	89	139	189	239	289	339	389	439	489
40	**90**	**140**	**190**	**240**	**290**	**340**	**390**	**440**	**490**
41	91	141	191	241	291	341	391	441	491
42	92	142	192	242	292	342	392	442	492
43	93	143	193	243	293	343	393	443	493
44	94	144	194	244	294	344	394	444	494
45	95	145	195	245	295	345	395	445	495
46	96	146	196	246	296	346	396	446	496
47	97	147	197	247	297	347	397	447	497
48	98	148	198	248	298	348	398	448	498
49	99	149	199	249	299	349	399	449	499
50	**100**	**150**	**200**	**250**	**300**	**350**	**400**	**450**	**500**

Remarks:

Figure I.- Timekeeper's check sheet

Doctor's First Report of Accident

Mine. ______ 19__

1. Name ______ Check No. ______
2. Address ______ Age ______ Sex ______
3. Occupation ______ Was this regular occupation? ______
   If not, state regular occupation ______ Married or Single ______
4. Number of Children under 16 ______ Nationality ______
5. Injured ______ 192__, ____ M
   Dr. Notified ______ 192__, ____ M
   Received ______ 192__, ____ M
6. First Aid by ______ At ______
7. Treatment by Dr. ______ At ______
8. Assistants ______
9. Interpreter? ______ Name and Address ______
10. STATEMENT OF INJURED PERSON AS TO MANNER IN WHICH INJURY WAS CAUSED
11. Nature and Extent of Injury In Detail ______
    See Diagram on Reverse Side
    Was Accident Fatal? ______
12. Treatment ______
13. Disposition of Patient ______
14. Probable Result ______
15. Probable period of disability ______
16. Previous condition and evidences of old injury ______
17. Insurance carried ______
18. Witnesses ______ Address ______
    Witnesses ______ Address ______
    Witnesses ______ Address ______
19. Remarks ______

______
Name of Attending Physician

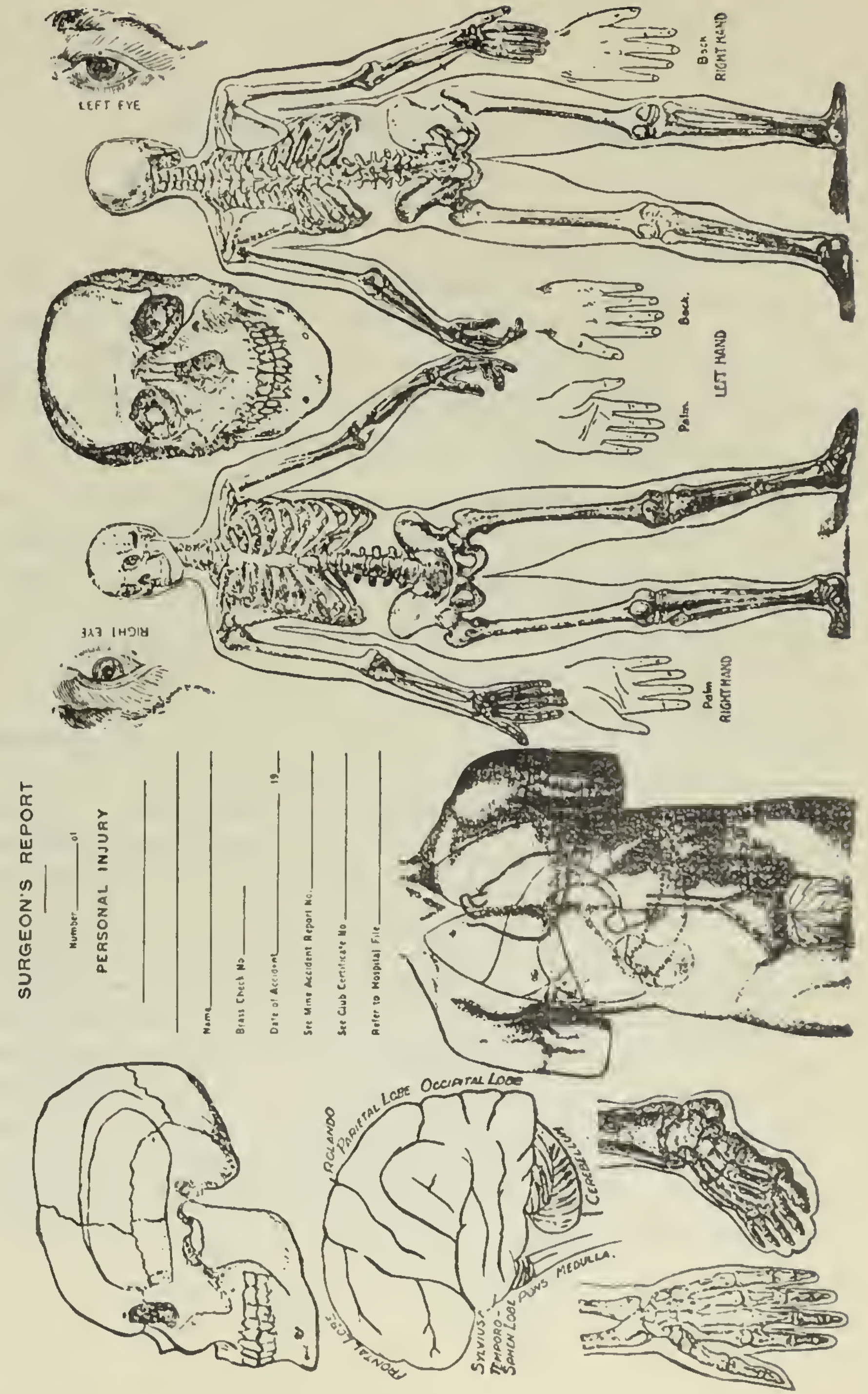


Figure 2.- Doctor's first report of accident

1M 1-30 Gault

Doctor's First Report of Accident

Employee (1) Name ______ B. C. No. ______

(2) Address ______ (St. No.) ______ (City or Town)

(3) Age ______ (4) Sex ______ (5) Married? ______

(6) Nationality ______ (7) Speak English? ______ (8) If not, what language ______

(9) Was injury due to wilful misconduct on part of employee ______ (10) What was the occupation of the person injured ______ (Have in mind work done)

(11) Length of experience here in this occupation ______ Elsewhere in this occupation ______ (12) Piece or day worker ______ (13) Working days per week ______ Wages per day ______ Per week ______

Time of accident (14) Date of accident ______ (Month) ______ (Day) ______ (Week) ______ M (15) Day of Week ______

(16) Hour of the day when injured began work ______

(17) How did the accident occur ______

Check the one or ones applicable.

Care of injured to date (18) At Hospital ______ (20) At Home ______

(19) At factory ______ (21) At whose expense ______

Nature and Extent of Injury (22) Was accident fatal ______ (Yes or No) (State exactly the part of person injured and nature of injury)

(23) Has the accident resulted in any permanent physical injury ______ (Yes or No) (24) Disability Total ______ (Yes or No)

Partial ______ (Yes or No) Probable duration in days ______

(25) Previous condition and evidences of old injury ______

(26) Attending Physicians

______ (Name) ______ (Address)

Remarks

Figure 3.- Doctor's first report of accident

Surgeon's Report of Accident

Date ______

Name ______ No. ______ Employer ______

Address ______ Occupation ______

Nationality ______ Age ______ Date of Birth ______

Single, Married, Widowed ______ No Family ______

Height ______ Weight ______ Hair ______ Eyes ______

Date of Accident ______ 19 ______ A. M. P. M. Place ______

Date of First Attention ______ 19 ______ A. M. P. M. Place ______

First Aid By ______ At ______

Treatment By ______ At ______

Injuries ______

Treatment ______

Probable length and extent of disability ______

Statement of injured person ______

Disposition of patient ______

How long employed ______

Evidence of previous injury ______

Witness of accident ______

Remarks ______

______ Surgeon

Figure 4.- Surgeon's report of accident

(77)

The Iron Company

REPORT OF PERSONAL INJURIES TO EMPLOYES

No. ______

______ Mine ______ 192 ___

EMPLOYE.

(1) Name ______ (2) Age ______ (3) Brass Check No. ______
(4) Nationality ______ (5) Does he understand English? ______ (6) Married ______
(7) Number of Children ______ (8) Street address ______
(9) Occupation ______ (10) How long employed at mine? ______

TIME, PLACE AND CAUSE OF ACCIDENT.

(11) Date and hour of Accident ______
(12) Place of accident in detail (if timbered, state condition; if not timbered, height of back) ______
(13) What caused the accident? (If fall of ground, state whether back or side or cave in) ______

SAFETY

(14) Did violation of any rule or safety regulation or improper use of safety device in any way contribute to the accident? Explain fully ______
(15) Could accident have been prevented? If so, how? ______
(16) Who instructed him as to the hazards of the work and when? ______
(17) Had he received a copy of the Safety Rules? ______

WITNESSES

(18) Brass Check numbers and names and addresses of witnesses ______

CARE OF INJURED PERSONS

(19) Was First Aid rendered? ______ (20) By whom? ______
(21) Where was he taken after accident? ______ (22) Attending surgeon ______
(23) Nature of injury ______

GENERAL.

(24) Statement of injured person, and person whom he blames, if anyone ______
(25) How long employed on work on which injured? ______
(26) Was accident due to negligence or want of care on part of injured, or any other person? If so, explain fully ______
(27) State when and by whom place was last inspected, prior to accident ______
(28) State total product lost each day on account of accident and time shut down ______

DEPENDENTS—TO BE ANSWERED IN CASE OF SERIOUS OR FATAL ACCIDENT

(29)* Names and addresses of wife, children under 16, and over 16 if mentally or physically incapacitated, and dates of birth of all such children ______
(30) Names and addresses of other dependents (Father, Mother, Grandmother, Brothers and Sisters), and extent of dependency ______

Noted ______ General Manager
Signed ______ Foreman
Signed ______ Captain
Date ______ Signed ______ Clerk.
Signed ______ Superintendent

*In Minnesota "Under 18 years of age."

(31) Was he engaged at his regular occupation? ______ (32) If not, state regular occupation ______
(33) Hour injured person began work that day ______
(34) What part of day's wages was paid for day of injury? (Give date) ______
(35) Number of working hours per day ______
(36) Number of days injured man regularly worked per week ______

No. ______

THE IRON CO.

REPORT OF PERSONAL INJURIES

Name ______
Mine ______
Dr. ______

Started work at ______ Mine
______ 19___

Formerly worked at ______ Mine.
(Give this information if possible.)

INSTRUCTIONS

This report must be made and sent to mine office immediately after the infliction of an injury to any person. ALL particulars must be noted fully. A separate blank must be filled out for each person injured or killed.

If any tonnage is lost by reason of injured person leaving work, the same must be reported under No. 28

General Manager

Voucher No.	FROM	TO	Amount Paid	Voucher No.	FROM	TO	Amount Paid

Figure 5.- Report of personal injuries to employees

2m 2 20 GPC 28201.

168

No. ______

THE IRON CO.

MINING DEPARTMENT

REPORT OF ACCIDENT TO EQUIPMENT

______ MINE ______ 192__

1. Accident to (Describe fully, showing manufacturer's name)	
Date and hour of occurrence.	
Day or night shift	
2. Location. If in shaft or level state at what point.	
3. Cause of accident	
Report fully any defects in material causing accident.	
When last inspected and by whom.	
In charge of	Occupation
Witness	
4. Length of time to repair	
Repaired by	
Commenced Repairing	Finished Repairing
Resumed Operations	
5. Cost of Repairs (Itemize charges, labor and supplies, use back of sheet if necessary for detail)	
	Total Cost
6. Product lost by accident. (Give dates and loss each day)	
7. General remarks (Any suggestions for the improvement of the plant tending to reduce accidents and consequent delays should be made)	
State if any personal injury was sustained and by whom	
Was accident caused through any violation of safety rules	

NOTED

______ Agent

______ 192

Signed ______ Mechanic Signed ______ Supt

Signed ______ Clerk

Report to be sent to central office promptly, in duplicate ... ary to be made for record at mine office

No. ______

THE IRON CO.

REPORT OF ACCIDENT TO EQUIPMENT

Name ______

Mine ______

Date ______

REMARKS

INSTRUCTIONS

1. This report must be made by the person in charge of equipment and sent to mine office as soon as possible after the occurrence of an accident where any additional information will be added and the report sent to central office in duplicate.

2. The information contained must be full and accurate, giving complete detail as to cause of accident, time delayed, etc.

3. This report must be made for all cases where any delay is caused and any expense for repairs is incurred by the equipment in all departments, such as Electrical Machinery, Boilers, Hoisting Machinery, Wire Rope, Skips or Cages, Compressors, Air Pipes, Pumping Machinery, Underground or Surface Tram Cars, Tracks, Engines, etc.

4. Attention is called to Section 7 calling for suggestions for the improvement of equipment tending to reduce accidents.

5. Immediately upon the occurrence of an accident the same must be reported by letter in duplicate to the Central Office. Report No. 169 to follow as soon as repairs are completed and all cost ascertained.

6. Any defective material must be carefully preserved for inspection by the Master Mechanic.

7. If any personal injury was sustained by employe or others in connection with this accident, the same must be fully reported giving Personal Injury report number (form 77).

Figure 6.- Report of accident to equipment

THE IRON COMPANY

MINE SAFETY COMMITTEE ACCIDENT REPORT

Mine____________________

1 Name of Injured____________________Accident Report No.______

2 Date and hour of Accident____________________

3 Cause of Accident____________________

4 The Following is recommended to prevent recurrence:____________________

Date____________________

____________________Chairman

Committee

Action of Superintendent on above report____________________

Superintendent

Figure 7.- Mine safety committee's accident report

MINING CO.

FOREMAN'S REPORT OF ACCIDENT

No.____Mine __________19___

Name____________________No.______

Date of Accident________________Hour______

How Accident Occurred:____________________

FOREMAN

Figure 8.- Foreman's report of accident

1M 1-31 G P Co

Foreman's Report of Accident

Date of Accident ______________________________ Hour ____________

Injured Person's Name ______________________ Age ______ Occupation ____________

Nationality ______________________________ Check No. ____________

Place of Accident, in detail ______________________________

____

Married ______________________________ Single ______________

Cause of Accident and how it happened ______________________________

____

____

____

____

Residence ______________________________

____

Nature and extent of injury ______________________________

____

____

Give names of all who saw the accident or who know anything about it ______________

____

____

____

____

____
Foreman

First-Aid Accident Report

Mine ______ *Time* ______ . *M.* ______ *19* ______

Name and No. ______ *Cant. No.* ______

Address ______________

Cause of Accident ______________

____

____

____

Nature of Injury ______________

____

How could accident have been prevented? ______________

____

First Aid rendered by ______________

Material used ______________

____

Signed ______________
FOREMAN

____

Certified Correct ______________
INJURED PERSON

(Use reverse side of report when necessary)

Figure 9.- Foreman's report of accident and first-aid accident report

1m. 7-18 G.P.C

FOREMAN'S ACCIDENT REPORT

Injured Person's Name ________ Brass Check No. ________

Date of Accident ________ Hour ________

Occupation ________ Party No. ________

Nationality ________ Married ________

Residence ________

Cause of Accident and How it Happened ________

Nature of Injury ________

Give names of all who saw or know anything about the accident ________

Had this man received any instructions? ________

If so what were they? ________

Did the injured man disobey any Safety Rule? ________

Did the injured man give notice of any defects? ________

Was injured notified of any danger or defect? ________

If so by whom? ________

Was First Aid Practiced in this case? ________

If so by whom? ________

Mine ________ Shaft ________ Level ________ Sub. ________

Foreman ________

NOTE: Please fill out and send to the Main Office.

F. 800. 12-21. 10M. GPN

MINING COMPANY

ACCIDENT REPORT

If accident is fatal or involves serious injury, telegraph or telephone immediately to General Claim Agent.

District........ District File No........

Mine........ Dept........

THE INJURED EMPLOYE—

1. Name........
2. Address........
3. Check No........ 4. Age........
5. Nationality........
6. Speaks English........
7. Usual Occupation........
8. Exact duties when injured........
9. Name the person who taught or instructed him........
10. ....Wages $........per day........
11. Tonnage, Time or Contract worker........
12. Hour began work that day........
13. Average working hours per day........
14. Average working days per week........
15. How long employed........
16. Married-Single-Widower........
17. If married, is he living with his family........
18. No. of children........

TIME AND PLACE OF ACCIDENT—

19. Calendar Date of Accident........Day of Week........Hour of Day........M
20. When did injured first begin to lose time—Date........Hour........
21. First day of Waiting Period........
22. Place of Accident........
23. Foreman general charge........ 24. Foreman immediate charge........

NATURE OF INJURY—(Obtain from Surgeon)........

25. Was First Aid rendered........ 26. By whom........
27. If infected by neglect, give date and hour when first given attention........
28. What reason given for neglect........
29. Where sent by Foreman and how........
30. Probable length of time off duty........ 31. Name of Surgeon........
2. Where sent by Surgeon........ 33. How........

THE CAUSE—

34. What was the man or gang of men doing and how did the accident happen........
35. Who is authority for answer to question 34........
36. To whom made........ In presence of........
37. Give names and check numbers of witnesses........

THE REMEDY—

38. In what way was a safety device or safety rule involved........
39. How could accident have been prevented........

Report made out........19.... by........ Foreman........ Noted........ Position........
Noted........ Superintendent

(For added space for any question use reverse side)

Figure 10.4 Foreman's accident report and company accident report

Record of Accidents

Accident No. .... Name of injured employee ....
Address .... Age .... Nationality .... Sex ....
Single, Married, Widowed or Divorced .... Occupation..
Department or branch of work .... Was this regular occupation ....
If not, state regular occupation ....
How long so employed .... Place of Birth ....
No. of children under 16 ....
Date of Accident ...., 19......
Hour of Accident ....
Hour injured began work ....
Was full wages paid for that day ....

Name and age of dependants

NAME	AGE	RELATIONSHIP

Daily wage employee was earning at time of accident $.. Weekly wage $ ....
Working hours per day .... Days worked per week ....
Place of accident in detail ....
Cause and manner of Accident ....
....
Nature of Injury in detail ....
....
Length of time lost by employee .... Date of return to work ....
Total amount Medical, Surgical and Hospital Cost $.................... Funeral Expense $....................
Name of Attending Physician .... Address ....
Name and address of Hospital to which Employee was taken ....
Date discharged by Physician .... Final result of injury ....

FIRST REPORT OF ACCIDENT		COMPENSATION AGREEMENT		COMPENSATION APPROVED BY BOARD		AMOUNT AWARDED BY BOARD			
Date	Form	Date	Form	Date	Form	Date	No of Weeks	Amount Per Week	Total Amount

Payments to be made to .... Address ....
Relationship ....

RECORD OF PAYMENTS

DATE OF PAYMENT	Week of Disability	Amount	Total Payments	Date Paid to	DATE OF PAYMENT	Week of Disability	Amount	Total Payments	Date Paid to	DATE OF PAYMENT	Week of Disability	Amount	Total Payments	Date Paid to

Figure 11.- Record of accidents

13-Rev. 7-29 I.O. 3700 THE IRON COMPANY (733)

MINING DEPARTMENT

REPORT OF SLIGHT ACCIDENT
NOT OVER ONE DAY'S TIME LOST

Mine .... Date of this report .... 19 ....

Name .... Occupation .... Brass Check No. ....

Age .... Address ....

Nationality .... Date and time of accident ....

Where working when accident occurred ....

Cause of accident ....

Nature of injury ....

Name, residence and occupation of eye witnesses. ....

Signed by .... Clerk

IMPORTANT NOTICE—Report on this blank all slight accidents on account of which an employe does not lose over one full day's time. If a man is off over one full day on account of an accident a report must be made on Form 77

Figure 12.- Report of slight accident

34-Rhs.-4-30-Lts. (718)

Final Certificate for Payment for Injuries

Acc't. Rep't. No. ....

Phy's. Rep't. No. 12221

.... Hospital .... 19....

THE IRON CO.

Mr. .... was unable to work

from .... to ....

on account of injury to .... received at .... Mine.

.... 19.... was discharged .... 19....

.... Attending Physician.

(Certificate to be made out by Attending Physician)

TIME STATEMENT

.... Time on ....
.... Time on ....
.... Time Clerk.

Approved by .... Mining Captain.

Accidents must be reported on Physician's Report before payment will be made.
When the injured man is ready to resume work this certificate must be sent at once to the General Office. The physician will make out this final certificate for payment at the same time that the card for injured men is signed by him for presentation to the Mine Clerk stating that the employe is ready for work

Figure 14.- Final certificate for payment for injuries

THE IRON CO. (718)

PHYSICIAN'S REPORT OF ACCIDENT

Acc't. Rep't. No. .... No. 12221

.... Hospital .... 19....

Name of injured person .... Residence ....

Nationality .... Place of birth .... Age ....

Married or single .... No. in family .... No. of children under 16 years ....

Date of Accident .... 19.... At .... Mine

Date examined .... Patient's statement of occurrence.

Description of injury ....

Disposition made of patient ....

Probable time under treatment ....

X-Ray ....

Remarks: ....

.... Attending Physician

This report to be sent, as soon as possible, to the General Office, Compensation Department, to be attached to accident report.

Figure 13.- Physician's report of accident

OLD EMPLOYEES

File No. ....

THE MINING COMPANY

PHYSICAL EXAMINATION

Name .... Address .... Check No. ....

Previously examined for .... Co. Date .... 19....

Age .... Speak English .... Read and write English ....

Vision and condition right eye .... left eye .... Conjunctivitis .... Trachoma ....

Hearing right ear .... left ear .... varicose veins ....

Hernia, right inguinal .... left inguinal .... ventral ....

Blood pressure .... systolic .... diastolic ....

Condition of heart ....

Urine appearance .... Sp Gr .... Sugar .... Albumin ....

Symptoms and history of epilepsy ....

Previous injuries ....

Deformities ....

REMARKS. ....

Recommendations ....

Date .... 192....

For THE MINING COMPANY,

.... M. D.

400 1-29 GPC

Figure 15.- Physical examination card for old employees

C. Form No. 1. 7M. 6-22.

Industrial Commission of Minnesota

612 Bremer Arcade
ST. PAUL

Case No.........................

Self Insurer's No.........................

ANSWER QUESTIONS FULLY

EMPLOYER'S FIRST REPORT OF INJURY
(Also to be used in Occupational Diseases)

(See General Instructions on back of this form)

EMPLOYER

NAME OF EMPLOYER— MINING COMPANY.
Office Address: Street and No.—
City or town—Duluth, County—St. Louis, State—Minnesota.
BUSINESS (State nature of business)—Iron Mining and Operations incidental thereto.
LOCATION of plant or place of work (if different from office)
Street and No.
City or town .... County .... State ....
DATE OF ACCIDENT .... 19 ... about ... o'clock ... noon ...

INSURANCE CARRIER

NAME OF INSURANCE CARRIER—(SELF INSURER).
General office address
Minnesota office address

INJURED PERSON

NAME OF INJURED EMPLOYEE
Post office address
Age ... Sex ... Married or single ... Speak English?
Occupation when injured
Was this regular occupation?
Piece or time worker ... No. working days per week
Wages per week ... Was board furnished?
Was room furnished? ... Value per week of board and room furnished $
Were wages paid weekly, semi-monthly or monthly
IF FATAL:
Date of death
Name and P. O. Address of dependent or friend of deceased

CAUSE OF INJURY

Describe fully how injury happened
Name of machine, tool or appliance in connection with which injury occurred (if any)

NATURE AND EXTENT OF INJURY

Give full and exact description of the injury
Was a member or part of member lost?
Will injury be likely to result in serious disfigurement?
Did injury cause loss of time? ... If "yes," when did lost time begin .... 19...
Has injured person returned to work? ... If "yes," what date? .... 19...
If not returned, give estimate of probable disability
Will any permanent disability probably result?

MEDICAL AND HOSPITAL

Did employer provide or authorize medical attention ... When
Name and address of attending physician (if any)
Name and address of hospital (if any)
Amount paid for medical and hospital service (if any) $

COMPENSATION LIABILITY

Is employer willing to pay compensation, subject to the conditions and limitations of Chapter 82, Laws 1921
(Yes or no) .... If not, is liability to pay compensation denied? ....
(Yes or no)

MINING COMPANY.

Dated .... 19...

Signed by........................

Official title—GENERAL CLAIM AGENT.

GENERAL INSTRUCTIONS

1. Report on this blank must be filled out and sent in within 48 hours in all fatal or serious cases; otherwise on the 7th day after injury or disability occurs.

2. If accident results fatally after this report is made, a further report of that fact must be filed.

3. This report should be filled out on typewriter or plainly written in ink.

4. Reports of injury must be made on forms prescribed by the Commission, with definite and complete answers to all questions, otherwise reports will be returned for correction.

5. If liability to pay compensation is not denied, employers should commence payment of compensation at the time and in the manner prescribed by Part 2 of the Compensation Law without the necessity of any agreement or order of the Commission, such payments to be made at the intervals when wages were payable as nearly as may be. (See Sec. 9, Chap. 82, Laws 1921.)

6. No compensation is allowed for the first week after disability begins, except that in the event incapacity for work continues for a period of four weeks from the day the disability begins, then compensation shall be computed from the commencement of such disability.

7. Evidence of compensation payments, in form of Receipts or duplicate copy of Drafts, must be filed with the Commission immediately following each payment, weekly, semi-monthly, or monthly, as the case may be.

8. All industrial accidents must be reported to the commission, whether compensable or not.
(See Chap. 359 Laws 1919)

Figure 16.- Employer's first report of injury

APPLICATION FOR EMPLOYMENT

No. Mine Check No.

Date Hired Date Settled Cause of Leaving

Name in Full Age Years Months

Post Office, Box No. Location Street, House No.

Nat'l'y Place Birth Date Birth

M'r'd Wid'r S'gle Name of Wife Address

Child'n No Ages

Other Dependents

Position Des'd Experience

Are You a Citizen of the United States

Have You Made Application For Citizen Papers

RECORD OF PREVIOUS EMPLOYMENT FOR 5 YEARS

COMPANY	CITY	CAPACITY	FROM	TO	CAUSE OF LEAVING

In Case of Serious Accident or Illness Notify the Following:

Address of Dependents in Old Country

Signed Applicant

(OVER)

MINING CO.

PHYSICAL EXAMINATION

Name Address Check No.

Age Speak English Read and write English

Vision and condition right eye left eye Trachoma

Hearing right ear left ear varicose veins

Hernia, right inguinal left inguinal ventral

Condition of heart Blood Pressure

Symptoms and history of epilepsy

Kidneys

Previous injuries

Deformities

Miscellaneous

Recommendations

Date 192 M. D.

File No. ....

New Employees THE MINING COMPANY

PHYSICAL EXAMINATION

Name .... Address .... Check No. ....

Previously examined for ....Co. Date .... 19 ...

Age .... Speak English .... Read and write English ....

Vision and condition right eye ....left eye .... Conjunctivitis .... Trachoma ....

Hearing right ear .... left ear .... varicose veins ....

Hernia, right inguinal .... left inguinal .... ventral ....

Blood pressure .... systolic .... diastolic ....

Condition of heart ....

Urine: appearance ....Sp. Gr. .... Sugar .... Albumin ....

Symptoms and history of epilepsy ....

Previous injuries ....

Deformities ....

EMARKS: ....

Recommendations ....

Date .... 192 .... For THE MINING COMPANY.

1000 9-29 CPC .... M. D.

Figure 17.- Application for employment, physical examination card, and physical examination card for new employees

Form A-12 Employer's No. (Do not fill in) Accident No. (Do not fill in)

INDUSTRIAL COMMISSION—FIRST REPORT OF ACCIDENT OR INDUSTRIAL DISEASE

(1) Date of report. Made out by. Position.

1. Employer

(2) Employer's name (Individual or firm name)

(3) Office address (Street No.) (City or town)

(4) Nature of Business (Goods produced, work done or kind of trade or transportation)

(5) Insured under compensation act in Insurance Co.

(6) Where did accident or exposure occur (if not at office address) (Street No.) (City or town)

2. Injured

(7) Name.

(8) Address (Street No.) (City or town)

(9) Age (Give date of birth, if injured is under 18) (10) Sex (11) Permit on file (if injured was minor under 17). (Yes or No)

(12) Speak English (Yes or No) (13) In what department or branch of work employed (If insured, state classification to which payroll is charged)

(14) Occupation (Kind of work done) (15) Length of experience with this employer

(16) Wages per week $ (including bonuses, etc.) (17) Hours per day. Days per week

3. Time of Accident

(18) Date of accident or first illness (Month, day, year) Last day worked (Month, day, year)

4. Cause of Accident or Disease

(19) Machine, tool or thing in connection with which accident or disease occurred (If machine indicate part)

(20) Was machine or part guarded? (Yes or No)

(21) Was guard properly attached at time of accident? (Yes or No) (22) If not, who removed it?

(23) In cases of industrial disease underline hazard to which exposed 1. Dust, gas, fumes. 2. Dirt. 3. Darkness. 4. Dampness or heat. 5. Bad air. 6. Fatigue or inactivity. 7. Poisons. 8. Infections.

(24) In case of industrial diseases describe method for control of hazards noted above

(25) Describe in full how accident occurred, or how employee was exposed to hazard (Immediate cause of injury or disease)

(26) What other conditions helped to cause accident or disease (Poor light, rough or slippery floors, overcrowding, defective ... (Home conditions, personal hygiene, etc.)

5. Nature and Extent of Injury

(27) Nature of injury or disease (State exactly the part of the person affected and the character of injury or disease ... If amputation give exact point)

(28) Did accident or disease result in death? (Yes or No) (29) Probable duration of injury or illness

(30) Attending physicians (Name) (Address)

(31) Dependents (give names of dependents in fatal cases only) Write this information on other side of this sheet according to following form:

(Name) (Relationship) (Age) (Address)

Figure 18.- Industrial commission; first report of accident or industrial disease

Industrial Commission of Wisconsin, Madison, Wis. Form A-10

SUPPLEMENTARY REPORT ON ACCIDENTS AND INDUSTRIAL DISEASES

(See rule at foot of page)

File No. E (Do not fill in) Accident No. (Do not fill in) Docket No. (Do not fill in)

(1) Employer's Name Address

(2) Name of Insurance Company Address of Adjuster

(3) Name of injured employe

(4) Address of injured employe Street and Number City or Town

(5) Date of accident or illness (6) Last day employe worked

(7) Nature of injury or illness This should be stated in every case

(8) Rate of wage per $\left\{\begin{matrix}\text{month}\\ \text{week}\end{matrix}\right\}$ $ Loss of wage (if disability is partial) $

(9) Compensation payments since last report Date $ Amount; Date $ Amount; Date $ Amount; Date $ Amount; Date $ Amount; covering in all the period from Date to Date weeks and days

(10) If payments were not made at the times required by law, state reason

FINAL REPORT ON ACCIDENTS

(To be filled out when final settlement is made in addition to the questions above)

(11) Date of final settlement (12) Date injured able to return to work

(13) Payments to compensate for injury Total $

As death benefit $ For permanent total disability $

For temporary total disability $ for weeks and days

For permanent partial disability $ Nature of disability (Follow schedule ... possible. If relative injury state percentage of disability)

For temporary partial disability $ from Date to Date weeks and days Loss of wages $ per day

(14) Medical treatment $ (15) Hospital treatment $ (16) Funeral expenses $

(17) For artificial members $ as follows Description

(18) Remarks

Rule 8. Employers or their insurers on all accidents which require a FIRST REPORT must:

1. Make a SUPPLEMENTARY REPORT, on this form on the fifteenth day following that on which the accident occurred.

2. Make another supplementary report immediately when payments are stopped for any reason. This report must be accompanied by an explanatory memorandum, if there is a dispute with the injured man.

3. Make a FINAL REPORT on this form when final payment of compensation has been made, which must be accompanied by (1) a copy of the final receipt signed by the injured employe, and (2) A REPORT FROM A PHYSICIAN IF THE DISABILITY EXCEEDS THREE WEEKS OR IF THERE IS ANY PERMANENT DISABILITY UNLESS THERE HAS BEEN A HEARING BEFORE THE COMMISSION

Figure 19.- Supplementary report on accidents and industrial diseases

Industrial Commission of Wisconsin, Form A-16

PHYSICIAN'S REPORT ON ACCIDENT OR INDUSTRIAL DISEASE

Name of Employe ________________________________

Name of Employer ________________________________

Date of accident or first illness ____________________ Date of amputation ____________

Date on which injured was able to return to work ________________________

Has accident or industrial disease resulted in any permanent disability? ________________

Describe physical mutilation or impairment resulting from accident or disease. (Where practical, make use of diagram on back of report.) ________________________________

____

____

____

____

Give date that healing period ended ________________________

If the accident caused injury to more than one of the constituent parts of the hand, state healing period separately for each injury ________________________

____

____

____

If amputation

1. Mark point of amputation on diagram ________________________
2. Has stump good, hardy pad or a tender pad? ________________________
3. If amputation was made between joints, state percentage of the functional value of several phalanx remaining ________________________

Remarks upon case ________________________________

____

____

Dated at ____________ this ______ day of ______________, 192____

Signature of physician ________________________________

Name of Case

I certify that I marked the above diagram as indicated

Name of Physician

Figure 20.- Physician's report on accident or industrial disease

Form No. E. (See Rule "E.") 12-29-30 5,000

STATE OF MICHIGAN
DEPARTMENT OF LABOR AND INDUSTRY
LANSING, MICHIGAN

SUPPLEMENTAL REPORT OF FATAL ACCIDENT

Make within fourteen (14) days after death of employe. (See Sub. (e), Sec. 17, Pt. 3, Law)

THIS FORM IS FURNISHED BY THE DEPARTMENT OF LABOR AND INDUSTRY. REPORTS MUST BE MADE ON THIS FORM. This report to be made by the employer to the Department of Labor and Industry within 14 days after the death of an injured employe.

1. Name of employer ____________________
2. Address of employer ____________________
3. Name of deceased employe ____________________
4. Nationality of deceased employe ____________________
5. Exact last address of deceased employe ____________________ (Including street, house number and post office)
6. DATE OF ACCIDENT WHICH RESULTED IN DEATH OF DECEASED EMPLOYE ____________________
7. Date of death of deceased employe ____________________
8. Did you furnish all medical aid required during final illness? ____________________
9. Amount of compensation (if any) paid to deceased employe before his death $ ____________________
10. Name and address of person who incurred the expense of burial of deceased employe ____________________
11. Name and address of person to whom any expense of final illness is due ____________________
12. Names, ages, relationship to deceased, extent of dependency, and address of ALL DEPENDENTS of deceased employe:

Names	Ages	Relationship to Deceased	Extent of Dependency (Total or Partial)	P. O. Address

Date of this report ____________________

(SIGNATURE OF EMPLOYER) ____________________ (To be signed in INK)

By ____________________

(Position of person signing) ____________________ (State clearly position, official or otherwise, with employer)

INSTRUCTIONS

This report is to be mailed to the Department of Labor and Industry at Lansing, Michigan, within 14 days after the death of an employe.

Figure 21.- Supplemental report of fatal accident

Form No. 19 10-19-29—1000

DEPARTMENT OF LABOR AND INDUSTRY

AGREEMENT TO PAY COMPENSATION TO DEPENDENTS ON ACCOUNT OF DEATH OF INJURED EMPLOYE

To be made and filed with the Department of Labor and Industry during the SECOND week after the death.
(See Sec. 17, Pt. 3, Law.)

THIS FORM IS FURNISHED BY THE DEPARTMENT OF LABOR AND INDUSTRY AND IS TO BE USED ONLY WHERE DEATH OF INJURED EMPLOYE RESULTS AND COMPENSATION IS PAYABLE TO DEPENDENTS.

THIS AGREEMENT, Made by, between, among and for the benefit of each and all of the dependents of the deceased employe hereinafter named, whose postoffice address ______________ hereinafter stated and ______________
(Name of employer, insurance company, or State Accident Fund)

WITNESSETH as follows:

IT IS AGREED, That the employer or insurance company signing this Agreement has reached an Agreement under the provisions of the Michigan Workmen's Compensation Law in regard to compensation or death benefits on account of the death of the deceased employe, ______________ while in the employ of ______________
(Name of employe)

____
(Name and address of employer)

and that the facts with reference to said matter are as follows:

1. DATE OF ACCIDENT ______________ 2. Hour of accident ______________
3. Cause of accident ______________
4. Describe fully the injuries resulting from the accident ______________

____

____

____

5. DATE OF DEATH OF EMPLOYE ______________

IT IS FURTHER AGREED, That the deceased employe left dependents wholly dependent upon him within the meaning of the Workmen's Compensation Law, as follows:

Names of Persons Wholly Dependent	Age of Dependent	Relationship of Dependent to Deceased	Amount per week Going to Said Dependent	P. O. Address of Dependent
1				
2				
3				
4				
5				
6				
7				
8				

IT IS FURTHER AGREED, That the death of the said employe on the date above named was caused by the accidental personal injury sustained by him on the date above named, and that his average weekly wage was $ ______________
([illegible])
and that the employer or insurance company signing this Agreement shall pay to the said dependents of the said employe who were wholly dependent upon him for support at the time of the injury, compensation at the rate of $ ______________
([illegible])
per week, for a period of ______ weeks from the ______________ day of ______________, A. D. 19____

IT IS FURTHER AGREED, That said deceased at the time of his death left dependents only partially dependent upon him for support within the meaning of the Workmen's Compensation Law, as follows:

Names of Persons Partially Dependent	Age of Dependent	Relationship of Dependent to Deceased	Amount per Week Going to Said Dependent	P. O. Address of Dependent
1				
2				
3				
4				
5				

IT IS FURTHER AGREED, That the employer or insurance company signing this Agreement shall pay to the said dependents of the said employe who were partially dependent upon him for support at the time of the injury, compensation at the rate of ______________ dollars per week for a period of ______________ weeks, from the ______________ day of ______________ A. D. 19____. Such payments to be made to each of the partial dependents in the amounts above set forth.

IT IS FURTHER AGREED, that this Agreement is made and entered into by the employer or insurance company signing the same, for the benefit of all persons dependent upon the deceased within the meaning of the Workmen's Compensation Law, whether wholly or partially, and each and all of the dependents of the deceased within the meaning of the Workmen's Compensation Law shall be considered as parties to this Agreement whether it be signed by them or not, and payments hereunder shall be made to each and all of them in accordance with the provisions of the Michigan Workmen's Compensation Law.

THIS AGREEMENT is made and submitted to the Department of Labor and Industry for its approval in accordance with the provisions of the Michigan Workmen's compensation Law.

Dated at ______________, this ______________ day of ______________ 19____

WITNESSES:

____

____

(Signatures MUST be in INK)

Said persons being all of the persons wholly dependennt upon the deceased.

____
(Name of employer, insurance company, or State Accident Fund in INK)

By ______________

(Position of person signing) ______________
(State clearly in INK position, official or otherwise, with employer or insurance company)

THIS GENERAL FORM IS MADE TO COVER ALL CASES. IT SHOULD BE FILLED OUT TO COVER THE FACTS OF EACH PARTICULAR CASE.

Figure 22.- Agreement to pay compensation to dependents on account of death of injured employee

Form No. C. (See Rule "C".) 1-15-30- 10M

DEPARTMENT OF LABOR AND INDUSTRY

AGREEMENT WITH INJURED EMPLOYE IN REGARD TO COMPENSATION

To be made and filed with the Department of Labor and Industry during the SECOND week after the accident. (See Sub. (c), Sec. 17, Pt. 3, Law.)

THIS FORM IS FURNISHED BY THE DEPARTMENT OF LABOR AND INDUSTRY. AGREEMENTS WITH INJURED EMPLOYE ON ANY OTHER FORM WILL NOT BE ACCEPTED.

We, ______________________________
(Name of Employe)

whose exact postoffice address is ______________________________
(Address of Employe)

and ______________________________
(Name of employer, insurance company, or State Accident Fund.)

have reached an agreement under the provisions of the Michigan Workmen's Compensation Act, in regard to compensation for the injury sustained by employe ______________________________

in the county of ______________________________, State of Michigan

while in the employ of ______________________________
(Name and address of employer.)

1. DATE OF ACCIDENT ______________ 2. Hour of accident ______________

3. Cause of accident ______________________________

4. IMPORTANT. Describe fully the injuries resulting from the accident ______________________________

____

____

____

____

The terms of the Agreement follow:

The employe's average weekly wage being $ ______________________________
(This is SIX times the daily wage, salary or emolument employe was earning at the time of injury.)

IT IS AGREED. That compensation be paid at the rate of $ ______________________________
(This is 66⅔% of his average weekly wage stated above, but not less than $7 nor more than $18 per week.)

per week, during ______________ Disability (of ______________ weeks)
(State whether total or partial.) (If loss of member results, state number of weeks on this line.)

The rate of compensation must not be changed except by supplemental agreement approved by the Commission or by an award. ALL in accordance with the provisions of the Michigan Workmen's Compensation Law.
IT IS FURTHER AGREED. That in case the said injured employe has suffered the loss of one or more members scheduled in Section ten, Part two of said act and is incapacitated in his employment at the end of the above period because of injuries other than the loss of said members as a result of said accident, he shall be entitled to compensation for Total or Partial Disability as the facts and law warrant at the end of said period.

Dated at ______________ this ______________ day of ______________, 19____

WITNESSES (Two witnesses to the signature of employe MUST sign in INK.)

____
(Signature of employe in INK.)

______________________________ ______________________________
(Name of employer, insurance company, or State Accident Fund in INK.)

______________________________ By ______________________________

(Position of person signing)
(State clearly in INK position, official or otherwise, with employer or insurance company.)

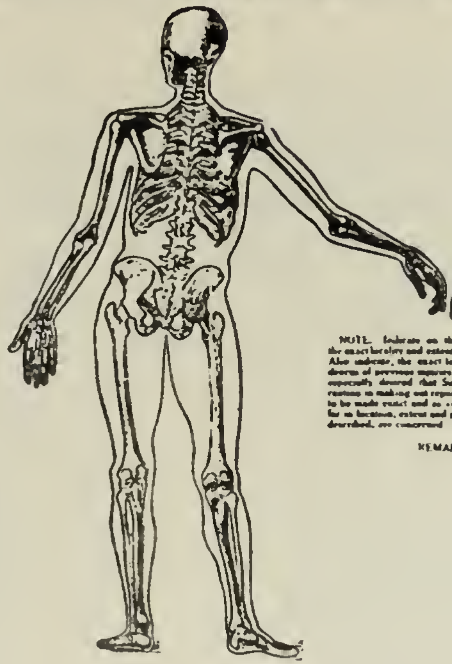
AGREEMENTS WITH INJURED EMPLOYES NOT ACCEPTED UNLESS EXECUTED IN COMPLIANCE WITH THIS FORM.
SEE REVERSE SIDE OF THIS BLANK FOR FURTHER INSTRUCTIONS.

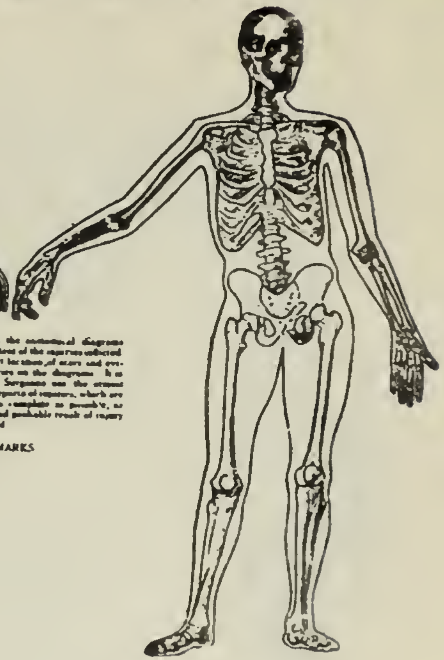
INSTRUCTIONS

1. This Agreement MUST be made and signed and filed in the office of the Department of Labor and Industry, at Lansing, during the SECOND week after the accident. The injured employe MUST be paid his first compensation IN ALL CASES on the fourteenth day after he is injured.

2. If this Agreement is NOT filed in the office of the Department of Labor and Industry, at Lansing, within fourteen days after the employe is injured, he will be sent blanks so that he may give notice of the accident to the employer and make formal claim for compensation.

In all cases of amputation, the diagram below should be used to designate the exact point of amputation, which must be marked and certified by the operating surgeon and him only. In cases of amputation of arm or leg surgeon must state exact distance below elbow or knee of such amputation.

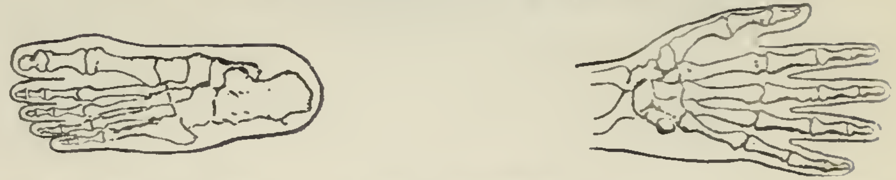




NOTE. Indicate on the anatomical diagrams the exact location and extent of the injuries inflicted. Also indicate the exact location of scars and evidence of previous injuries on the diagrams. It is especially desired that Surgeons use the utmost care in making out reports of injuries, which are to be made exact and as complete as possible, so far as location, extent and probable result of injury described, are concerned.

REMARKS

When injuries are on Hand or Foot use diagram below



I hereby certify that I marked the above diagram on ______________ 19____, and that said
(Date of marking.)

marking correctly indicates the amputation(s) made upon ______________________________
(Name of injured employe.)

on 19 and that the remarks above, if any, are in my handwriting.

____
(Signature of Operating Surgeon.)

Figure 23.- Agreement with injured employee in regard to compensation

Form No. B. (See Rule "B") 6-4-20—100 (4A)

STATE OF MICHIGAN
DEPARTMENT OF LABOR AND INDUSTRY
LANSING, MICHIGAN

REPORT OF COMPENSABLE ACCIDENT

Make this on the 8th day after accident. (See Sub. (b), Sec. 17, Pt. 3, Law.)

THIS FORM IS FURNISHED BY THE DEPARTMENT OF LABOR AND INDUSTRY. REPORTS MUST BE MADE ON THIS FORM.

To be made on the 8th day after the accident by the employer to the Department of Labor and Industry in cases involving loss of member, or death, or disability continuing more than 7 days.

1. Name of employer
2. Address of employer
3. Nature of business
4. Location of plant or place of work where accident occurred
5. Name of injured employe
6. Address of injured employe (Including street, house number and post office)
7. Occupation of injured employe
8. Department or branch of work
9. Was this regular occupation?
10. If not, state regular occupation
11. How long so employed
12. Place of birth
13. Sex
14. Age
15. Single, married, widowed, divorced
16. Number of children under 16 years
17. DATE OF ACCIDENT
18. Hour of accident
19. Hour injured began work that day
20. Was full wage paid for day of injury (if not, that day must be included in waiting period)?
21. Daily wage, salary or emolument employe was earning at time of accidental injury
22. Average weekly wage (This is six (6) times the amount given under item 21) $
23. Working hours per day
24. Days worked per week
25. Place of accident in detail
26. Cause and manner of accident
27. Nature and extent of injury in detail
28. Name of attending physician
29. Exact address of attending physician
30. Name and address of hospital to which injured employe was taken (if any)
31. Name of insurance company carrying employer's risk on date of accident

Date of this report

(SIGNATURE OF EMPLOYER) (To be signed in INK.)

By

(Position of person signing) (State clearly position, official or otherwise, with employer.)

INSTRUCTIONS

This report is to be made by the employer to the DEPARTMENT OF LABOR AND INDUSTRY on the 8th day following the accident when the employe has not returned to work within 7 days after the accident, and is ALWAYS to be made in cases of loss of member or death.

In case the accident causes the loss of a member, state exactly what, and the precise point of amputation; for example, the index finger of the right hand at the second joint, or the left arm at the elbow, the right eye, etc.

If the employe is entitled to compensation, an Agreement should be made and filed with the Department of Labor and Industry between the 8th day after the accident and the 14th day after the accident.

This report to be mailed to the Department of Labor and Industry, Lansing, Michigan.

EVERY QUESTION ON THIS BLANK MUST BE ANSWERED FULLY—INCOMPLETE OR INDEFINITE REPORTS WILL NOT BE ACCEPTED.

Figure 24.- Report of compensable accident

Form No. A (See Rule "A") 1-07-[illegible]

STATE OF MICHIGAN
DEPARTMENT OF LABOR AND INDUSTRY
LANSING MICHIGAN

REPORT OF NON-COMPENSABLE ACCIDENT

Make in all cases on 8th day after Accident (See Sub (a), Sec 17, Pt. 3, Law.)

THIS FORM IS FURNISHED BY THE DEPARTMENT OF LABOR AND INDUSTRY REPORTS MUST BE MADE ON THIS FORM

This report is to be made by the employer to the Department of Labor and Industry on the 8th day after every accident on account of which the employe loses no time, or little time, or returns to work within 7 days and has no compensation due. It is not to be filed in cases of loss of member or death, or disability continuing more than 7 days. If the injured employe has not returned to work send Form No. B instead of this.

1. *Name of Employer*
2. *Address of Employer*
3. *Nature of business*
4. *Location of plant or place where accident occurred*
5. *Name of employe*
6. *Address of employe (including street, house number and postoffice)*
7. *Occupation of employe*
8. *Age*
9. *Sex*
10. *DATE OF ACCIDENT*
11. *Hour of accident*
12. *Nature and cause of injury*
13. *Length of time lost by employe*
14. *DATE OF RETURN TO WORK*
15. *Amount of medical, surgical and hospital expenses (if any) paid up to date of report $*
16. *Name of attending physician*
17. *Exact address of attending physician*
18. *Name and address of hospital to which injured employe was taken (if any)*
19. *Name of insurance company carrying employer's risk on date of accident*

If said employe hereafter becomes entitled to compensation on account of said accident a report will be immediately made by as under Rule "B"

Date of this report

(SIGNATURE OF EMPLOYER) (To be signed in INK.)

INSTRUCTIONS

Every Item of This Report Must Be Fully Filled Out.

Only non-compensable accidents are to be reported on this Form. All compensable accidents must be reported under rule "B" on Form B. THIS REPORT must be signed by someone who can verify the statement that the employe has returned to work, viz: foreman, timekeeper, etc.

Mail this to the Department of Labor and Industry Lansing, Michigan.

By

(Position of person signing) (State clearly position, official or otherwise, with employer.)

Figure 25.- Report of noncompensable accident

Form No. 11

STATE OF MICHIGAN

DEPARTMENT OF LABOR AND INDUSTRY

Lansing, Michigan

RECEIPT ON ACCOUNT OF COMPENSATION

(See Sub. (d) Sec. 17, Pt. 3, Law)

Receipt for the ....week of disability.

THIS FORM IS FURNISHED BY THE DEPARTMENT OF LABOR AND INDUSTRY. Receipts Must be Made on This Form.

.... vs. ....

(Name of injured employe.) (Name of employer.)

RECEIVED OF ....

(Name of employer or insurer.)

the sum of..dollars and ....cents

being the proportion of the weekly wages of my*..

FROM the..day of ....19....

TO the ....day of 19

(INCLUSIVE), due under the Michigan Workmen's Compensation Law, subject to review by the Department of Labor and Industry, SAID ACCIDENT OCCURRING ON THE....................day of............................19

while injured employe was in the employ of..

(Name of employer.)

$..................................

Date of this receipt...19........

WITNESSES (Two witnesses to the signature of employe or dependents MUST sign below in INK.)

....

(Signature of employe or dependent in INK.)

....

(Street and house number.—This MUST be filled in.)

....

(City or village and state.—This MUST be filled in.)

RECEIPTS NOT ACCEPTED UNLESS WITNESSED AND SIGNED IN INK.

*Self, Husband or other, as the case may be.

Form No. 11 (See Rule "D")

STATE OF MICHIGAN

DEPARTMENT OF LABOR AND INDUSTRY

Lansing, Michigan

RECEIPT ON ACCOUNT OF COMPENSATION

(See Sub. (d), Sec. 17, Pt. 3, Law.)

Receipt for the..........................week of disability.

THIS FORM IS FURNISHED BY THE DEPARTMENT OF LABOR AND INDUSTRY. Receipts Must be Made on This Form.

.... vs. ....

(Name of injured employe.) (Name of employer.)

RECEIVED OF ....

(Name of employer, insurance company or Commissioner of Insurance.)

the sum of..dollars and ....cents

being the proportion of the weekly wages of my*..

FROM the..day of..19........

TO the ....day of..19........

(INCLUSIVE), due under the Michigan Workmen's Compensation Law, subject to review by the Department of Labor and Industry, SAID ACCIDENT OCCURRING ON THE........................day of..................................19........

while injured employe was in the employ of..

(Name of employer)

$..

Date of this receipt...19........

WITNESSES (Two witnesses to the signature of employe or dependents MUST sign below in INK.)

....

(Signature of employe or dependent in INK

....

(Street and house number.—This MUST be filled in.)

....

(City or village and state. This MUST be filled in.)

RECEIPTS NOT ACCEPTED UNLESS WITNESSED AND SIGNED IN INK

This receipt is to be mailed to the Department of Labor and Industry, Lansing, Michigan, AS SOON AS IT IS SIGNED

*Self, Husband or other, as the case may be

Figure 26.- Receipt on account of compensation

Form No. 1A. (See Rule "11.")

STATE OF MICHIGAN
DEPARTMENT OF LABOR AND INDUSTRY
LANSING, MICHIGAN

FINAL REPORT OF ACCIDENT

This report MUST accompany the Settlement Receipt, which MUST be filed with the DEPARTMENT OF LABOR AND INDUSTRY, when the final payment is made.

THIS FORM IS FURNISHED BY THE DEPARTMENT OF LABOR AND INDUSTRY. REPORTS MUST BE MADE ON THIS FORM.

1. Name of employer......
2. Address of employer......
3. Name of person injured......
4. Occupation of injured employe......
5. Weekly wages of injured employe $......
(This is six times the daily wage, salary or emolument he is earning at the time of the accidental injury.)
6. Total amount of compensation paid $......
7. Number of weeks compensation paid......
8. Total medical, surgical and hospital cost $......
(Exclusive of services of company surgeon.)
9. Date of final payment......
10. DATE OF ACCIDENT...... 11. Date of return to work......
(Give exact date when he DID return to work.)
12. Date of this report......

(SIGNATURE) ......
(To be signed in INK by person making this report.)

(Position) ......
(State clearly position, official or otherwise, with employer or insurer.)

INSTRUCTIONS

EVERY ITEM OF THE ABOVE MUST BE FULLY FILLED OUT

Incomplete reports will not be accepted. This report to be mailed to the DEPARTMENT OF LABOR AND INDUSTRY by the employer or insurer with the Settlement Receipt.

Figure 27.- Final report of accident

1000 2-30 Qualit

RECEIPT ON ACCOUNT OF COMPENSATION FOR INJURY

...... 193 .....

RECEIVED of The Mining Company, the sum of ...... Dollars - $ ......
being the proportion of my weekly wages for the ...... week of disability, from ...... 19 .... to ...... 19 ...., inclusive, due under the Wisconsin Workmens Compensation Law, subject to the review of the Industrial Commission of Wisconsin, said accident occurring on ...... 19 ...., while in the employ of The Mining Co.

Witness: ......

...... Employee

...... Address

Figure 28.- Receipt on account of compensation for injury

______, 192__

Received of ______ **MINING COMPANY** the sum of

______ DOLLARS

In full settlement and satisfaction of a claim by me against said **Mining Company**, *based upon an injury received while in its employ on the ______ day of ______ 192__, which caused a disability from said date until the ______ day of ______ 192__. Said sum is inclusive of all compensation and medical treatment claimed or demanded by me under Chapter 50, Laws of 1911, known as the Workmen's Compensation Act.*

Dated at ______ the ______ day of ______ 192__

WITNESSED BY

______ ______
Signature of Employee.

______

Computation ______

______

______

Figure 29.- Settlement receipt

The______________Mining Company

Frequency and Severity Table

For Period Ending__________192___

	UNDERGROUND										
	No. Shaft Month	Season	No. Shaft Month	Season	No. Shaft Month	Season	No. Shaft Month	Season	No. Shaft Month	Season	
Total Number of Shifts Worked											
Total Number of Lost Time Accidents.											
Total Number of Days Lost.											
Frequency Rate of Lost Time Accidents per 1000 Shifts.											
Severity Rate of Days Lost per 1000 Shifts.											

	GENERAL SURFACE										
	Electrical and Mechanical		Blacksmith and Drill Shop		Surface						
	Month	Season	Month	Season	Month	Season	Month	Season	Month	Season	
Total Number of Shifts Worked.											
Total Number of Lost Time Accidents											
Total Number of Days Lost.											
Frequency Rate of Lost Time Accidents per 1000 Shifts.											
Severity Rate of Days Lost per 1000 Shifts.											

Combined Frequency and Severity Table

	Total Underground		Total Electrical, Mechanical, Blacksmith & Drill Shop Surface		Grand Total		Compensation Payments		
	Month	Season	Month	Season	Month	Season	Month	Season	
Total Number of Shifts Worked.									
Total Number of Lost Time Accidents									
Total Number of Days Lost									
Frequency Rate of Lost Time Accidents per 1000 Shifts									
Severity Rate of Days Lost per 1000 Shifts									
Total Number of Fatal Accidents.									
Frequency Rate per 1000 Tons of Ore Produced.									

Figure 30.- Frequency and severity table

THE MINING CO.

Name Check No.	Accident No.
Address (Street, Village, Street)	First Report to Industrial Commission.
Nationality	
Age Married, Single, Widower Wages per Day, $	First Report to Co.
Wages per Week, $ Rate of Compensation per Week, $	
Date and Hour	
How did Accident Occur?	Final Report to Industrial Commission.
Nature of Injury.	
	Final Report to Co.,
Mine Location of Accident.	Doctor's Supplementary and Final Report
	to the Industrial Commission of Wis.
Shift Boss Fatal	Date of Doctor's Discharge.

SPECIAL DOCTOR'S FEES

Date	Name of Doctor	Amount
	TOTAL	

SPECIAL MEDICAL EQUIPMENT

Date	Equipment	Amount
	TOTAL	

AMBULANCE FEES

Date	From To	Amount
	TOTAL	

TRAVELING EXPENSES

Date	From To	Amount
	TOTAL	

FUNERAL EXPENSES

Date	To Whom Paid	Amount
	TOTAL	

INDUSTRIAL COMMISSION AWARD FOR DEATH

Date	Awarded to	Amount

PERMANENT OR PARTIAL PER. DISABILITY

Date	Disability	Amount

COMPENSATION PAYMENTS Rate $

Date	Week	From	To	Days	Weeks	Amount	Date	Week	From	To	Days	Weeks	Amount
							TOTAL						

Figure 31.- Summary of compensation data

RECORD OF ACCIDENT FOR MONTH OF ______________ 192___

MINE	SHAFT	EMPLOYEE	Occu-pation	Name of Foreman	Date of Accident	Accident Number	DESCRIPTION OF ACCIDENT IN DETAIL	Un avoidable	Care lessness	First Aid Given	Number of Days Lost	

Figure 32.- Monthly record of accidents

NAME

PAYMENTS AS PER AWARD OF INDUSTRIAL COMMISSION FOR DEATH OR DISABILITY

Date	Week	Amount	Date	Week	Amount	Date	Week	Amount	Date	Week	Amount	Date	Week	Amount
												TOTAL		

Disabled From	to	=	Days, or	Weeks	Days
Disabled From	to	=	Days, or	Weeks	Days

TOTAL DAYS DISABLED			REMARKS:
Special Doctor Fees			
Special Medical Equipment.			
Ambulance Fees			
Traveling Expenses			
Industrial Commission Award for Death			
Award for Permanent or Partial Permanent Disability.			
Compensation Payments			
Legal Expense			
TOTAL COST			

Figure 33.- Record of payments as per award of industrial commission for death or disability

I. C. 6568

October, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XXX

BY

Frederick W. Lee

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]

No. 30

Compiled by Frederick W. Lee[2]

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6568."

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1. GRAVITATIONAL METHODS

(413) CORRECTION TOPOGRAPHIQUE RELATIVE A L'EMPLOI DE LA BALANCE DE TORSION

(TOPOGRAPHIC CORRECTION WITH RELATION TO THE USE OF THE TORSION BALANCE)

By C. Alexanian

Congrès International des Mines, Liège, 1930, June, Mémoire No. 36, pp.269-274.

The author describes a new and general method of topographic correction. The article consists of the following three headings:

1. Introduction.
2. Principle of the new method.
3. Mathematical consideration of the method.

The translation of the author's abstract reads as follows:

> The author proposes to apply for each section of topographic configuration around the torsion balance a well-defined hyperbolic paraboloid (minimum surface of the second degree); this makes possible a complete integration of expressions relative to this correction without the necessity of applying approximations admitted so far. For this purpose the formula used by Eötvös is only a particular case of that proposed by the author.

Two figures and a number of tables which may be used by prospectors are added.--W. Ayvazoglou.

(414) GRAVIMÉTRIE DES FORMATIONS PÉTROLIFÈRES

(GRAVIMETRY OF PETROLEUM FORMATIONS)

By Arnaldo Belluigi

La Revue Pétrolifère, Paris, No. 417, 1931, pp. 405-411.

A detailed mathematical discussion on the effect of gravity on petroleum formations is given.

Graphical determination of gravimetric influences is explained.--W. Ayvazoglou.

(415) SCHWEREMESSUNGEN UND GEOLOGIE VON MITTELASIEN

(GRAVITY MEASUREMENTS AND GEOLOGY OF CENTRAL ASIA)

By Pavel Savitsky

Gerlands Beiträge zur Geophysik, Leipzig, vol. 30, No. 3/4, 1931, pp. 277-280.

By comparing the gravitation-stations with the geological maps of Central Asia the limits of areas with regard to gravimetrical data may be fixed as follows:

1. The Ferghana Valley is characterized by considerable negative anomalies, which can be attributed to the lowering of the rock layers caused by the compression of the earth crust owing to the pushing up of masses (Alaiden).

2. The region of the Karakul Lakes (Pamir) has positive anomaly, characteristic of plateaus.

3. Kalan-Wamar represents the most considerable negative anomaly in Central Asia.

4. The region of the Issyk-kul Lakes has positive anomaly, probably due to the great mass surplus below the bottom of the lake.

5. The Choresmic positive anomaly is caused by the presence of massive crystalline rocks.

6. The Krasnovodsk positive anomaly may be attributed to the extension of the intrusive and mass crystalline rocks.

The comparison of the value and of the sign of the anomalies with the age of the geological formation shows the prevalence of positive anomalies in the regions of the old and intrusive crystalline rocks and the predominance of negative anomalies in the regions with rock of later ages.--Author's abstract translated by W. Ayvazoglou.

(416) INTERPRÉTATION DES ANOMALIES GRAVIMÉTRIQUES ET MAGNÉTIQUES DE L'ALSACE

(INTERPRETATION OF GRAVIMETRIC AND MAGNETIC ANOMALIES IN ALSACE)

By J. Jung and C. Alexanian

Annales des Combustibles liquides, Paris, vol. 6, No. 1, 1931, pp. 43-58.

In the introduction to this article the authors observe that the study of gravimetric and magnetic anomalies of the earth's crust has been directed during the last years into two almost independent courses: The development of local geophysical prospecting for ore deposits, and the establishment of large nets of stations for obtaining data on physical properties of large areas and even of whole continents. In comparing these two courses with local

topographic survey and a geodetic triangulation, the authors consider it evident that in the future the development of studies of regional geophysics will be brought into accord with the local geophysical investigation and will serve as a basis for this latter prospecting. As these conditions exist already, more or less, in Alsace the authors have chosen this country for examining the problem of interpretation of the gravimetric and magnetic anomalies of this area.

The question is discussed under the following headings:

1. Gravimetric and magnetic anomalies: Gravimetrical measurements used; measurement of density of rocks; corrections applied to the calculation of anomalies; new values of gravimetric anomalies (given in a table); new map of gravimetric isanomalies; magnetic measurements used; magnetic anomalies (shown in a table); a map of isanomalies of Z.

2. Comparison and significance of anomalies: Reason for gravimetric anomalies caused at great depth; anomalies of the supracrystalline origin in the Fossé Rhénan (Rhine Valley); anomalies of the infracrystalline origin in the Vosges; significance of magnetic anomalies; significance of magnetic and gravimetric anomalies; reason for infracrystalline anomalies; depth of basic infracrystalline; relation between the infracrystalline anomalies and the geology of the surface.

Four maps complete the article.--W. Ayvazoglou.

2. MAGNETIC METHODS

(417) ÜBER DIE ERMITTLUNG TEKTONISCHER LINIEN MITTELS DER MAGNETISCHEN FELDWAAGE IN GEBIETEN GERINGER UNTERSCHIEDE DER MAGNETISCHEN VERTIKALINTENSITÄT, IM BESONDEREN IN NORDDEUTSCHLAND

(ON THE DETERMINATION OF TECTONIC LINES BY MEANS OF A MAGNETIC FIELD BALANCE IN REGIONS WITH SMALL MAGNETIC VERTICAL-INTENSITY DIFFERENCES, ESPECIALLY IN NORTH GERMANY)

By E. Kohl

Kali und verwandte Salze, Berlin, vol. 25, Nos. 14, 15, and 16, 1931, pp. 209-215, 225-230, and 241-243

Based on a series of examples, described and illustrated in this article, Kohl draws the following conclusions:

> Summing up the results of a series of investigations, there is no doubt that the determination of the tectonic elements and lines in regions with small magnetic intensity differences, as for example in North Germany, is absolutely possible by using the present-day magnetic instruments. From a practical viewpoint this is important for the discovery of ore deposits connected partly with the tectonic lines or lying in the neighborhood of these lines. On the other hand, the magnetic method of prospecting may be of value to geology for making clear the tectonic

conditions in such areas the surface maps of which cannot serve the purpose owing to the coverings of younger periods and if there are no mines, as often happens in north Germany.

Although a scientist often refuses to accept the lines resulting from the isonomal maps, considering them to be of insufficient value for his purposes, they still may give him valuable indications.

Distrust is often shown with regard to the accuracy of measurements attainable. The reason is because the present-day instruments are much affected by temperature variations, thus great care is required to obtain the necessary accuracy of measurements. The new magnetic systems with improved temperature compensation, now under construction, will perhaps make it possible to attain the maximum accuracy more easily or even to increase the accuracy of measurements.--Author's abstract translated by W. Ayvazoglou.

3. SEISMIC METHODS

(418) SEISMIC PROSPECTING

By L. Don Leet

The Military Engineer, Washington, D. C., vol. 23, No. 130, 1931, pp. 326-330.

In a brief historical sketch the author mentions the first experimental determinations of elastic-wave velocities made by R. Mallet (1856) and the development of these experiments since 1913 (Reginald A. Fessenden). The discoveries of salt domes in Texas, Louisiana, Oklahoma, and other States of the United States by seismic prospecting are noticed.

Basic theory of seismic prospecting, explanation of terms and formulas expressing the velocities of the P-waves and S-waves are given.

The two next sections of the article deal with "Straight-line time-distance graphs" and "Curved time-distance graphs."

In discussing field procedure and large-scale prospecting, the author gives some figures of the cost of operating a standard seismic troop in the Gulf Coast salt-dome area ($15,000 to $20,000 per month, the acreage covered in reconnaissance shooting being from 150,000 to 300,000 acres). Although the seismic method is so far used mainly for discovering salt domes, one company established last summer that by contouring the surface of a nepheline syenite batholith in Arkansas they could effectively prospect for bauxite, the ore of aluminum.

The predicted depth was computed as 218 feet ± 6 feet. The drill encountered syenite at 220½ feet. The seismographic computations, incidentally, showed also that there was a sudden change in material at 55 feet. At approximately that depth, the drill reached the bottom of a loose sand and encountered a wet, sticky clay. Therefore, there is no reason why the method can not, as a matter of regular production routine, locate similar abrupt

velocity discontinuities with an error of less than 5 per cent. The possibility of application of the method in prospecting dam sites and building foundations is mentioned also.

In the last section, "Present status," the author says:

> Seismic prospecting has developed with amazing rapidity, but its growth has been neither healthy nor normal.... It has reached a stage where it is imperative that some broad programs of university research be devoted to a solution of its problems, whether immediate financial returns from their solution seem likely or not.
>
> It must be recognized that seismic prospecting is not a separate and distinct branch. It is a tool for the geologist, and only as such can it attain any satisfactory degree of usefulness.--W. Ayvazoglou.

(419) SEISMOGRAPH WORK WITH EXISTING WELLS

By Burton McCollum and Wilton W. LaRue

The Oil and Gas Journal, Tulsa, Okla., vol. 30, No. 5, 1931, pp. 24, 81-82.

The authors explain some of the recent seismic methods for exploration of the territory surrounding the existent wells for very deep-seated domes. The principle of the method, consisting of running a detector of seismic waves to the bottom of a wildcat well, is explained.

In describing the practical utilization of the method the authors believe that in view of a very large number of wells now existing around various salt domes in the Gulf Coast, this method can be made very effective in exploring this region for domes to a substantially greater depth than has heretofore been possible by the usual surface method of operation. A diagram illustrates the method for deep exploration of regions surrounding known domes; diagrams also show the use of a well for profiling the mushroomed flank of a salt dome, for determining the position of a fault plane at various depths, and of exploring the side of a dome opposite a well for mushrooming.

A typical record made with the special seismic wave detector in a well at a depth of 6,690 feet and a shooting distance of 29,490 feet is given in a figure.--W. Ayvazoglou.

(420) THE FORMATION OF LOVE WAVES (QUERWELLEN) IN A TWO-LAYER CRUST

By Harold Jeffreys

Gerlands Beiträge zur Geophysik, Leipzig, vol. 30, No. 3/4, 1931, pp. 336-350.

The generation of S H and Love waves from an impulsive source is considered. It is found that the disturbance transmitted through the lower layer consists of a series of overlapping pulses, each starting at an instant

corresponding to the time of transmission of a pulse that has undergone an integral number of reflections in the upper layer. The recovery after each pulse leads to a train of waves, the superposition of which gives the Love waves.--Author's abstract.

(421) SEISMOGRAMMFORMEN UND VORGÄNGE IM HERDGEBIET

(FORMS OF SEISMOGRAMS AND PROCESSES IN THE REGION OF THE EPICENTER)

By G. Krumbach

Gerlands Beiträge zur Geophysik, Leipzig, vol. 30, No. 3/4, 1931, pp. 351-365.

From a practical example, a Kamchatka earthquake, the author shows that in certain epicenter regions very characteristic forms of seismograms appear even within great periods of time and large regions of stations. To explain this phenomenon the author examines the influence of the processes in the region of the epicenter in general and tries to coordinate the results obtained with the opinions on the origin of earthquakes.--Author's abstract translated by W. Ayvazoglou.

(422) ÜBER OBERFLÄCHENWELLEN

(ON SURFACE WAVES)

By Tokunosuke Itō

Gerlands Beiträge zur Geophsik, Leipzig, vol. 30, No. 3/4, 1931, pp. 366-407.

The author presents a detailed mathematical discussion on the following items:

1. Deduction of fundamental relations in the visco-electric media.
2. Rayleigh-waves in visco-elastic media.
3. Love-waves in visco-elastic media.
4. Rayleigh waves in case of cooscillation of a retarding earth's crust layer in visco-elastic media.
5. Surface shearing waves in case of cooscillation of a retarding earth's crust layer in vicso-elastic media.
6. The influence of the overlying layer upon Rayleigh waves.
7. Dispersion of surface shearing waves with layers lying one above the other in visco-elastic media.--W. Ayvazoglou.

(423) ON THE TRANSMISSION OF SEISMIC WAVES ON THE BOTTOM SURFACE OF AN OCEAN

By Katsutada Sezawa

Bulletin of the Earthquake Research Institute, Tokyo, vol. 9, No. 2, 1931, pp. 115-142.

1. The viscosity of water gives no influence upon the transmission of Rayleigh-type waves on the bottom surface of an ocean.

2. The compressibility of water is effective only on the movement of the superposed water, but not on the motion of the ground.

3. The pressure of the water in the sense of forming surface water waves is of a great importance in the transmission of the seismic waves.

4. The velocity of harmonic waves changes with the wave length, and this velocity is quick or slow according as the wave length is long or short. There is no definite critical velocity for harmonic waves of an infinite extent.

5. In a two-dimensional transmission of waves, the azimuthal component of the displacement of the solid as well as of the water becomes quiescent as the distance from the epicenter is increased.

6. For long waves, the ratio of the horizontal component of the displacement to the vertical component is larger than that of the ordinary Rayleigh waves, while this ratio is smaller than that of the latter for short waves.

7. The motion of the water particle is mainly of a vertical type.

8. However concentrated the original disturbance may be, the transmitted waves are of a dispersed form. As the distance from the epicenter is increased, this tendency becomes more distinct.

9. The dispersed waves are apparently of an oscillatory type, having a tail with gradually decreasing periods and amplitudes.

10. The leading part of the dispersed waves is propagated with the velocity of the ordinary Rayleigh waves.

11. The phase of the leading part changes as the waves proceed to a long distance.

12. The wave lengths of the oscillatory part become larger and larger and the forms of the waves are more and more flattened in the course of the progression of the disturbance.

13. The disturbed portion has various group velocities, depending upon the harmonic elements involved in that portion. The maximum of the group velocities is the same as that of Rayleigh waves, and this maximum velocity corresponds with the harmonic element of an infinitely long wave length.

14. The phase velocity is also variable for different parts of the disturbed waves.

15. When the effects of gravity and of water are omitted, the transmitted waves due to a concentrated disturbance are of a gradually subsiding type without oscillation.--Author's abstract.

(424) ETUDE PRELIMINAIRE SUR L'ACCELERATION DES SEISMES

(PRELIMINARY STUDIES ON THE ACCELERATION OF SEISMIC WAVES)

By Mishio Ishimoto

Bulletin of the Earthquake Research Institute, Tokyo, vol. 9, No. 2, 1931, pp. 159-166.

Ishimoto describes a new apparatus, constructed by him, assigned for studying seismic phenomena. Studies made by the author were limited to establishing characteristics of shocks from the viewpoint of acceleration. The apparatus was installed in the basement of the Institute and observations were made during a period of about four months.

Results of 24 observations are shown in a table and the reproduction of some typical seismograms is given. The author draws the conclusion that from the viewpoint of acceleration some new conceptions of seismic movements may be established; but they must be studied by using another method of registration, as the photographic method is, in the opinion of the author, unsatisfactory for this purpose.--W. Ayvazoglou.

4. ELECTRICAL METHODS

(425) ÜBER DIE BERECHNUNG VON DEFORMIERTEN ELEKTROMAGNETISCHEN FELDERN IN DER GEOLEKTRISCHEN PROSPEKTION

(ON THE CALCULATION OF DISTORTED ELECTROMAGNETIC FIELDS IN GEOELECTRICAL PROSPECTING)

By A. Belluigi

Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 1, No. 4, 1931, pp. 363-372.

The magnetic field of two horizontal, balanced point charges on the surface of an infinite semispace is to be calculated. An interference for the lines of force is also arranged in the semispace. First, it is shown how the current distribution in the semispace is influenced by the interference. The field is then resolved into tubes of current, the deformation of the current potential is calculated, and from this the change in the tubes of current. From this change, according to Biot-Savart's law and by means of such graphic methods as used in gravimetrics, one derives the interfered, vertical components of the magnetic field; figure 1 gives the current strength of the individual tubes as ordinates; R : 1 as function of the abscissas. In Figure 2 the radii of the cross sections, the centers of which lie on the X-axis and through which 1/10, 1/20. ..... of the total current passes at intervals, are given as ordinates; the distance of the tubes of current from the center between the point charges is represented by the abscissa. Figure 3 gives the contribution of the various current lines to the components of the strength of the magnetic field.--Author's abstract.

(426) SULL' USO DI ELETTRODI DISSIMILI NELLA PROSPEZIONE GEOELETTRICA DEL SOTTOSUOLO

(ON THE USE OF DISSIMILAR ELECTRODES IN THE GEOELECTRICAL PROSPECTING OF THE SUBSOIL)

By Arnaldo Belluigi

Bollettino della Societa Geologica Italiana, Rome, vol. 50, No. 1, 1931, pp. 57-62.

Physical interpretation of the results of geoelectrical prospecting with alternating current by using dissimilar electrodes is explained.

A practical case which may frequently occur in the study of oil-bearing fields is discussed.--W. Ayvazoglou.

(427) EARTH RESISTIVITY SURVEYING

By G. F. Tagg

Engineering and Mining Journal, New York, vol. 131, No. 7, 1931, pp. 325-326.

A brief description is given of the earth resistivity method of geophysical surveying according to the system of measuring the specific resistance of homogeneous earth, described in the U. S. Bureau of Standards paper entitled "A method of measuring earth resistivity," by F. Wenner.

The interpretation of the results by the three different methods, (1) the empirical method, (2) the resistivity map method, and (3) the method of theoretical analysis, is explained.--W. Ayvazoglou.

(428) RESULTS OF ELECTRICAL RESISTIVITY AND ELECTRICAL INDUCTION MEASUREMENTS AT ABANA MINE, QUEBEC, CANADA

By E. Vernon Potter

U. S. Bureau of Mines, Technical Paper 501, Washington, 1931, 28 pp.

Contents:

Location and geology.
Mineralogy.
Acknowledgments.

Electrical resistivity surveys: Introduction; principle of methods; description of apparatus; precautions in using Megger; corrections for resistance in potential circuit; correction curves; plotting results; effect upon resistivity of geological discontinuity; choice of lines of observation; field observations.

Locating mineral deposits by alternating-current methods: Preliminary discussion; elliptical polarization; artificially produced fields; types of alternating current fields; field apparatus; circuit arrangement method of comparing major and minor axes of ellipse; field layout; interpretation of observations; determination of depth; sources of error; results of tests; comparison of methods.

The author discusses the theory as well as methods of surveying, illustrating the article with a series of figures. In comparing the two methods the author says:

> The two methods described in this paper are based on widely different principles, and each has its own special field of application. The resistivity method is necessarily slow, but very detailed results are obtainable from its correct use. The alternating-current method, however, is good for rapid work, but does not reveal the details, such as depth, slope, etc., nearly so well. A combination of two such methods should be very useful in making a survey over new country, the alternating current method being used for a preliminary survey and the resistivity method for a detailed survey in interesting points. Needless to say, both methods discriminate between differently conducting bodies which may or may not represent mineralization. These surveys are of value in districts in which, from a geological viewpoint, mineralizations are likely to occur.

Potter's paper is completed by F. W. Lee's article, "Explanation of some factors associated with induction method." The question is examined under the following items: Preliminary discussion; locus of magnetic field; mathematical relation between currents and magnetic fields; interrelation of parameters; specifications of measuring circuit; mathematical relations in measuring circuit; interpretation. Lee's summary of the article reads as follows:

> Summarized on very broad principles, the exciting loop carrying an alternating current may be considered a magnetic shell, the strength of which varies as the current changes in the wire. The direction and intensity of the field may be computed from elementary electrical formulas. This alternating magnetic shell energizes other magnetic shells in the ground by magnetic induction. If these other shells have sufficient strength they may be located by comparing their strength and direction to that of the main exciting shell. If the configuration of these detected shells have the proper geological forms or shapes for a certain district in that they may be associated with proper mineralization they should be more carefully investigated. On the other hand, erratic results may indicate conductors arising from water courses and other unconformities.

The bibliography on this subject is rather extensive and reference is here made to the Index to Geophysical Abstracts 1 to 20, prepared by the Geophysical Section of the U. S. Bureau of Mines and issued as Information Circular 6438.--W. Ayvazoglou.

5. RADIOACTIVE METHODS

(429) OBSERVATIONS ON THE PENETRATING RADIATION IN THE ANTARCTIC

By Kerr Grant

Nature, London, vol. 127, No. 3216, 1931, p. 924.

The author describes the observations upon the variation of intensity of the penetrating of cosmic radiation with latitude carried out by the British-Australian-New Zealand Antarctic Research Expedition during the recent summer cruise of the "Discovery," November, 1930, to March, 1931.

The apparatus employed was a Geiger-Müller electron tube counter, with single-stage amplifier, relay, and automatically recording chromograph.

Counts were obtained over a region ranging from Hobart to Adelie Land - that is, over a range of geographical latitude from 43° S. to 68° S.

The mean value (6.3 per minute) of all counts on the voyage is identical within the limits of experimental error with the value (6.1) obtained for a count of four hours in the physics laboratory of the University of Adelaide. The result of the observations thus tends to confirm those of Bothe and Kohlhörster in the North Atlantic, of Corlin at Abisco, and of Millikan at Churchill in Canada, in showing that the intensity of penetrating radiation does not vary to any considerable extent with magnetic latitude even within 250 miles of a magnetic pole.--W. Ayvazoglou.

(430) SUR LA DÉFINITION DE LA PÉRIODE DU POLONIUM EN DIVERS POINTS DE L'U. R. S. S.

(ON THE DEFINITION OF THE PERIOD OF POLONIUM AT DIFFERENT PLACES IN THE U. S. S. R.)

By M. L. Bogoiavlensky

Le Journal de Physique et le Radium, Paris, vol. 10, No. 9, 1929, pp. 321-328.

Owing to the fact that the considerable differences in the definition of the radioactive constant of polonium and of its period can not be explained merely by the inaccuracy of measurements, the experiments carried out by Bogoiavlensky and described in this article were made after a very careful preparation of polonium. After a brief description of the means used for preparing polonium, of the electromagnetic installation and of the technique of determining the period of polonium during the experiments, Bogoiavlensky gives the results of his measurements carried out in 18 different places in the U.S.S.R. in a table.

According to his calculations, the mean error of observations reached 8.3 per cent. In analyzing the errors caused by measurements the author draws the conclusion that the considerable changes in various places evidently depend on

local causes, and supposes that the content of radioactive substances in rocks greatly influences the value of the period. This value increases with the increase of the quantity of radioactive substances and is smaller in case of the decrease of this quantity.

A map showing the places in which observations were made is added.

Bogoiavlensky's article is followed by "remarks," written by Mme. P. Curie in which she says that, based on some considerations, it is necessary to make a more exact verification of the results published in this article.

In an additional note Bogoiavlensky admits that the remarks made by Mme. Curie have already been taken into consideration for a new series of experiments. --W. Ayvazoglou.

(431) ÉTUDE DE LA VITESSE DE DÉSINTÉGRATION DU POLONIUM EN DIVERS LIEUX

(STUDY OF THE VELOCITY OF DISINTEGRATION OF POLONIUM IN DIFFERENT PLACES)

By M. L. Bogoiavlensky

Le Journal de Physique et le Radium, Paris, vol. 2, No. 1, 1929, pp. 12-19.

In this article Bogoiavlensky gives the results of experiments carried out in continuation of those described in the article above. Three series of experiments for the determination of the constant of polonium were made: One in the southern part of the U.S.S.R. (42 points), one in the region of Leningrad (18 points), and one in the vicinity of Leningrad, in Dietskoe Selo (3 points). The results are given in three tables. They show that the variations are considerably smaller than those obtained during the experiments made in the previous year. This proves the existence of errors in previous investigations. Among them the author mentions the error caused by the standardized size of piece of polonium used.

Maps showing the investigated points, provided with figures of the results, are added.--W. Ayvazoglou.

6. GEOTHERMAL METHODS

(432) PROCÉDÉ DE PROSPECTION GEOTHERMIQUE

(GEOTHERMAL METHOD OF PROSPECTION)

By C. L. Alexanian

Extract from, Traité pratique de prospection géophysique (Practical treatise on geophysical prospecting), by C. L. Alexanian. Librairie Polytechnique Ch. Béranger, 15, Rue des Saints-Pères, Paris, 6.

Values of geothermal gradients are given by the author for the following countries:

Meters

	Meters
Pribram (Czechoslovakia)	67
Sabountchy (Oil region near Baku, U.S.S.R.).	25
Machôlles (Limagne).	14
Pechelbronn (Alsace)	14 to 21
Los Angeles (California)	14
Kutzenhausen (Alsace).	7

According to Koenigsberger the normal geothermal gradient in case of horizontal layers is about 34 meters. This value varies greatly in case of inclined or vertical layers.

The possible reasons for the great local anomalies of the geothermal gradient are given by Alexanian as caused by:

1. Deposits originated by heat processes or of volcanic origin.
2. The distribution of heat due to the displacement of various substances in the earth's crust, such as hot springs, mineral waters, and gas.
3. Deposits produced by heat caused by chemical reactions or deposits containing radioactive substances.
4. Thermal conductivity of rocks and tectonic displacements (heat produced by friction).

A list of thermal conductivity of rocks is given.

The last two paragraphs of the article contain a brief description of methods for measuring the temperature of the soil and a method for calculating the temperature gradient.—W. Ayvazoglou.

(433) GEOTHERMAL VARIATIONS IN COALINGA AREA, FRESNO COUNTY, CALIFORNIA

By Anders J. Carlson

Bulletin of the American Association of Petroleum Geologists, Houston, Tex., vol. 15, No. 7, 1931, pp. 829-836.

Variations of isothermal elevations with respect to structure are developed from temperature measurements in 56 wells of the Coalinga area. The comparative trend of isotherms, geologic strata, and ground surface is shown on two vertical sections through the anticlinal structure of the Eastside field. The results indicate that rock temperatures in this vicinity are controlled chiefly by surface topography and thickness of sediments. Definite correlation between relative temperatures and the oil-bearing structure is not evident.—Author's abstract.

7. UNCLASSIFIED METHODS

(434) PRINCIPLES AND PRACTICE OF GEOPHYSICAL PROSPECTING

By A. B. Broughton Edge and T. H. Laby

Report of the Imperial Geophysical Experimental Survey, London, 1931.

The book contains XIV + 380 pages with 261 illustrations in the text. It is published by the Cambridge University Press (London, Fetter Lane, E. C. 4). Price, 15 s. net.

The form in which this work of the Survey has been prepared for publication is more in the nature of a manual or textbook of geophysical prospecting, illustrated by field results, than of a formal report.

The volume is divided into two parts.

Contents:

Part I. Chapter 1. Introduction: Objects of the I. G. E. S. investigations; selection of survey areas in Australia; organization of survey and field work.

Chapter 2. Electrical methods: Principles and applications; surface potential methods (spontaneous polarization, resistivity and resistivity surveys, equipotential line, A. C. potential ratio); electromagnetic methods (Bieler-Watson, high-frequency).

Chapter 3. Electrical survey in Australia: New South Wales; Victoria; Tasmania; South Australia; Western Australia; Queensland.

Chapter 4. Gravimetric methods: Principles and applications; instruments; field procedure; gravimetric survey in Australia; conclusions.

Chapter 5. Magnetic methods: Principles and applications; instruments; field procedure; magnetic survey in Australia.

Chapter 6. Seismic methods: Principles and applications; seismometers; field procedure; seismic investigations in Australia.

Part II. Chapters 7 to 11.

The electrical, gravimetric, magnetic, and seismic methods described in Part I are dealt with in greater detail. Full descriptions are given of all instruments used by the Survey and of the procedure followed in carrying out field investigations. The methods are also considered from the theoretical point of view.—W. Ayvazoglou.

(435) ANCORA SULLE CARATTERISTICHE FISICHE DELLA PIANURA MARGINALE APPENNINICA MODENESE

(MORE ON PHYSICAL CHARACTERISTICS OF THE APPENNINIAN BORDER PLAIN OF MODENA)

By A. Belluigi

Rendiconti della R. Accademia Nazionale dei Lincei, Rome, vol. 12, No. 3-4, 1930, pp. 112-114.

In a previous article on the physical characteristics of the Appenninian border plain of Modena (see Geophys. Abs. 23, p. 80), Belluigi drew from the data obtained the conclusion that gypsum or similar minerals must be deposited at a certain depth in this region.

In this article Belluigi discusses briefly some considerations supporting the conclusions made in the first article.--W. Ayvazoglou.

(436) DISCRIMINATION IN APPLYING GEOPHYSICS

By Sherwin F. Kelly

Mining and Metallurgy, New York, vol. 12, No. 293, 1931, pp. 239-241.

Dissatisfaction with the achievements of geophysics, due often to a misconception of its capabilities and limitations, or to a poor application of its technique, or to a faulty perspective as to its proper place in geological work, caused the writing of this article as the author finds it imperative to help in understanding past achievements and failures, and through that to formulate workable programs for future progress by critical examination and correction of past records. Kelly expresses the hope that those in possession of information will engage in cooperation in providing the material which would be valuable in such a compilation. The article does not give any original contribution to geophysical science or art. The author discusses the question raised by him under the following headings: (1) Basic assumption, (2) lessons of hindsight, (3) valuable information, and (4) comparative study.--W. Ayvazoglou.

(437) GEOPHYSICS IN EXPLORATION: PROSPECT AND RETROSPECT

By Sherwin F. Kelly

Engineering and Mining Journal, New York, vol. 131, No. 1, 1931, pp. 11-12.

The purpose of this article is to clarify the question with respect to just what is the role geophysics should play in present-day exploration work.

The three distinct stages of the attitude toward, for example, electrical exploration (skepticism, over-optimism, indifference), are discussed.

A considerable list of literature dealing with this question is added.--W. Ayvazoglou.

(438) GEOPHYSICS IN THE MINING SCHOOLS

By V. Gavrilovich Gabriel

Engineering and Mining Journal, New York, vol. 131, No. 7, 1931, p. 333.

In this short article the author expresses his opinion on studying geophysics in universities and schools and recommends some ways by which, he hopes, the unnecessary expenditures of money by schools could be avoided and the enthusiastic students of geophysics may be saved from disappointment.--W.Ayvazoglou

(439) THE SCIENCE OF GEOPHYSICS

By B. Dunstan

Queensland Government Mining Journal, Queensland, vol. 32, Nos. 370, 371, 372, pp. 95-96, 146-148, 186-187.

After a brief discussion on the practice of geophysical prospecting, Dunstan gives a table of geophysical methods, somewhat similar to that published some years ago by Dr. Heiland of the Colorado School of Mines, which is built from the point of view of the geologist, primarily based on geological and mineral features, and showing, approximately, the geophysical methods to be adopted in the examination of certain mineral deposits, irregular rock masses, bedded structures, and dynamical features.

Some information is added on the restrictions to be applied when using the methods, and the determinations which it is difficult or impossible to accomplish. No attempt is made to describe in detail the geophysical methods used for the purpose of mineral prospecting, but simply an outline of these methods is given.

The methods are divided into six main groups, comprising electrical, gravimetric, magnetic, seismic, radioactive, and geothermal methods.--W. Ayvazoglou.

(440) PROGRESS OF GEOPHYSICAL PROSPECTING STUDIES

Editorial Note

Queensland Government Mining Journal, Brisbane, vol. 32, No. 370, 1931, p. 121.

In the course of studies of the use of geophysical methods in prospecting for minerals, work is being conducted by the United States Bureau of Mines, Department of Commerce, in the oil districts of Kentucky with a view to checking further the results of electrical investigations. The bureau is making progress on the devising of seismic calibrating apparatus, as well as on the improvement of torsion-balance instruments by developing a new method of accurately recording small angles.--W. Ayvazoglou.

(441) ÉTUDES GÉOPHYSIQUES APPLIQUÉES AUX RECHERCHES MINIÈRES

(GEOPHYSICAL STUDIES APPLIED TO MINING INVESTIGATIONS)

By Arnaldo Zabelli

L'Écho des Mines et de la Métallurgie, Paris, vol. 59, No. 3058, 1931, pp. 196-200.

In the beginning of this article Zabelli mentions the initiative taken by the United States Bureau of Mines in collecting and publishing the results of mining exploration by geophysical methods of prospecting and puts the question as to how the results obtained from this investigation can best be applied to the mineral industry in general, and how, for example, the following problems could be solved:

1. To establish in a scientific way to what degree there are appreciable links between the indications obtained by instruments and physical properties (electrical, magnetic, radioactive, thermic, electromagnetic, etc.) of minerals hidden in the soil.

2. To specify the importance of various geophysical methods of prospecting concerning the place of each of them which they must occupy in the industry of mining exploration.

3. To determine the best direction to be given to further investigatic s.

4. To find the best means for eliminating those unfit prospectors and inventors who, owing to their failures, only delay the development in the application of geophysics to prospecting for ores.

Based on a few examples given in this article the author concludes that under the actual knowledge of geophysical proceedings it is not possible to establish with certainty either the methods which may be applied in certain cases nor to classify them according to their degree of suporiority.--
W. Ayvazoglou.

(442) DIE POLARISATION DES HIMMELSLICHTES UND IHRE ZUSAMMENHÄNGE MIT ANDEREN METEOROLOGISCHEN ELEMENTEN

(SKY POLARIZATION AND ITS RELATIONS TO THE OTHER METEOROLOGICAL ELEMENTS)

By W. Smosarski

Gerlands Beiträge zur Geophysik, Leipzig, vol. 30, No. 3/4, 1931, pp. 408-424.

The annual variation of the sky polarization at noon and at sunset is discussed, taking into consideration the three components of the diffuse light-intensity. Then the relations are examined which have been observed between the simultaneous changes of the sky polarization, the intensity of solar radiation, the water-vapor tension, the electrical potential, the electrical conductivity of the air and other meteorological elements.

The relations seem to depend on the seasons of the year and also on the degree of polarization.--Author's abstract.

(443) LIMITATIONS OF GEOPHYSICAL METHODS IN OIL PROSPECTING

Editorial Note

Queensland Government Mining Journal, Brisbane, vol. 32, No. 373, 1931, p. 230.

Conditions of geophysical prospecting for oil are discussed, based on Lowe's article, "Geological Prospecting for Oil Still in Research Stage," published in National Petroleum News, vol. 21, No. 47, 1929, pp. 59-63 (see Geophys. Abs. 10, 1930, p. 24).--W. Ayvazoglou.

(444) GEOPHYSICAL PROSPECTING

Editorial Note

New York Times, N. Y., Monday, August 3, 1931, p. 32.

The note says that according to a Canadian Pacific Railway bulletin, further tests will be made on geophysical prospecting during this year by the following three Government field parties:

1. A. S. Eve and D. A. Keys of the McGill University will, in collaboration with the U. S. Bureau of Mines, conclude investigations started last year into the behavior of high-frequency radio waves underground (Mammoth Cave).

2. L. Gilchrist of the University of Toronto will continue research in the field with a variety of electrical and magnetic methods for the purpose of ascertaining their serviceableness in the search for concealed mineral deposits (chromite of Thetford, Quebec, magnetite of Bristol, Quebec, and pyrite of Calabogie, Ontario).

3. The Dominion Observatory and the Geological Survey are collaborating in research in gravitational methods of studying underground conditions. A. H. Miller of the Dominion Observatory started work late last year with torsion balance to ascertain its use in determining faults.

Results already accomplished in the survey of Canadian mineral resources are briefly mentioned.--W. Ayvazoglou.

(445) GEOPHYSICAL TESTS IN THE RHINE VALLEY

By R. P. Reichenbach and H. Bertram Bateman

The Mining Magazine, London, vol. 44, No. 5, 1931, pp. 280-285.

The authors describe the investigation of structural conditions in the Rhine Valley carried out by gravimetric, seismic, and radioactive methods of geophysical prospecting. Calculated values for gradients and values of curvature, as well as values resulting from the measurements, are shown in a figure.

From this figure it can be seen that the calculated values of the gradients for the geological profile underneath coincide with the measured values to within ± 1 unit. In case of the values of curvature the difference between the calculated and the measured curve amounts to about 25 units; the measured curve lies higher than the calculated one.

The results of the gravimetric survey in this area show that the bedrock rises from south to north and also from east to west. Only an approximate estimate of the depth could be given by gravimetric survey.

Seismic measurements carried out in the Lower Rhine basin southwest of Cologne serve as a further example of the elucidation of structural conditions.

The results of seismic survey are shown in accompanying diagrams and it can be seen that the seismic measurements were able to determine a remarkable number of faults of varying throw which dip northeastwards. At the same time the seismic measurements were able to determine the thickness of the Tertiary deposits.

Lastly, the radioactivity test measurements made at Hennef-on Sieg are briefly referred to. The results of these measurements can be seen from the curve of the measurement values shown in a figure.

The authors conclude that the results of these different geophysical surveys demonstrate that the practical geophysical methods now available, if expertly applied, can give very valuable data, not only on the geological conditions of an area from the mining point of view, but also from the purely geological standpoint.--W. Ayvazoglou.

(446) COMPTES RENDUS DE L'ASSEMBLÉE DE STOCKHOLM

(SECTION DE MAGNÉTISME ET ÉLECTRICITÉ TERRESTRES)

(PROCEEDINGS OF THE MEETING IN STOCKHOLM. SECTION OF TERRESTRIAL MAGNETISM AND ELECTRICITY. 15-23 AUGUST, 1930)

Bulletin 8, Union Géodesique et Géophysique International. Published by Ch. Maurain, 1931, 479 pp. Paris V, Les Presses Universitaires de France.

This volume includes the proceedings of the Section of Terrestrial Magnetism and Electricity of the International Geodetic and Geophysical Union held in Stockholm, Sweden, from August 15 to 23, 1930.

Detailed reports on the progress of magnetic-survey work and investigations were received from the following countries. They are given in Part II of this volume.

Australia. General report by J. M. Baldwin.

Canada. Report on observatory work at Agincourt and Meanook, by W. E. W. Jackson. Magnetic survey work in Canada by the Dominion Observatory, Ottawa, by C. A. French. Magnetic work accomplished by the Topographic Survey of Canada in 1927-1929, by F. H. Peters. Section of Terrestrial Magnetism and Electricity, National Committee of Canada.

Denmark. Report on the works on terrestrial magnetism in Denmark and Greenland, by D. la Cour.

Finland. Progress of work in terrestrial magnetism in 1928-1930, by J. Keränen.

France. Report by the secretary of the French section of magnetism and terrestrial electricity, by E. Mathias. Regular observations and publications concerning terrestrial magnetism, by Ch. Maurain.

Germany. Report on the earth-magnetic work in Germany, by A. Nippoldt.

Great Britain. Atmospheric electricity at Key Observatory. Work in terrestrial magnetism at Scottish observatories. Report of the work done in terrestrial magnetism and atmospheric electricity at Greenwich and Abinger, by F. W. Dyson. Note on the work in terrestrial magnètism done at Stonyhurst College Observatory, by E. D. O'Connor. Ordnance Survey, Magnetic work. Work on terrestrial magnetism and atmospheric electricity, by S. Chapman and collaborators. Work in terrestrial magnetism carried out by A. Crichton Mitchell.

Italy. Note on the work executed in Italy on magnetic field and terrestrial magnetism in 1928-1929.

Japan. National report of Japan, by A. Tanakadate.

Mexico. Report of the magnetic station of the Astronomic Observatory of Tacubaya, by Joaquim Gallo.

Netherlands and Netherlands East Indies. Terrestrial magnetism in the Netherlands and Netherlands East Indies.

Norway. Report of the magnetic works in Norway, 1927-1929, by O. Krogness. Report from the Norwegian Institute for Cosmic Physics, in charge of (a) The Northlight Observatory, Tromsö; (b) The Magnetic Bureau, Bergen, by L. Vegard. Report regarding the first year's work at the Northlight Observatory, Tromsö, by Leiv Harang.

Poland. Magnetic survey in Poland, by St. Kalinowski. Report on the magnetic investigations of the geophysical institute of Lwow University during the years 1928-29, by Edward Stenz and Henryk Orkisz.

Portugal. Report on the terrestrial magnetism in Portugal, by A. Ferraz de Carvalho. Works on the terrestrial magnetism in the Azores, by J. Agostinho.

Siam. Magnetic observations, by the Director, Royal Survey Department.

Spain. A brief report by M. Luis Rodes.

Sweden. Magnetic work accomplished by the Hydrographic Office of Sweden from 1927-1930, by Gustaf Reinius. Summary report of the earth-magnetic investigations carried out in the mainland of Sweden, by Kurt Mollin.

Switzerland. Works on terrestrial magnetism in Switzerland from 1927 to 1930, by P. L. Mercanton. Magnetic survey of Switzerland by W. Brückmann.

United States. Report of work of section of terrestrial magnetism and electricity of the American Geophysical Union, 1928-1930, by J. A. Fleming and Harlan W. Fisk. Progress-report of Naval Research Laboratory, by E. D. Almy. Condensed summary work of the geophysical section, U. S. Bureau of Mines, by Scott Turner. Report on work of Radio Laboratory, Newton Centre, Massachusetts, by Greenleaf W. Pickard. Progress-report of Mount Wilson Observatory, by W. S. Adams. Report by the Department of Terrestrial Magnetism, Carnegie Institution of Washington.-

Part III contains reports concerning special matter as resulting from decisions taken at the meeting in Prague, such as:

Report of Commission on magnetic characterization, by A. Crichton-Mitchell.

Magnetic characterization, suggestion from Stonyhurst College Observatory, by J. P. Rowland.

The Atlas of Auroras.

Relations between the International Meteorologic Organization and the International Geodetic and Geophysical Union.

Terminology and symbols. Report by Ch. Maurin. Observations by Cabrera van Dijk and Fleming.-

Part IV. Communication on various subjects:

A. General note of the Department of Terrestrial Magnetism, Carnegie Institution of Washington (comments on the agenda for the Stockholm meeting) contains the following articles: Magnetic characterization of days, studies of sudden commencement of magnetic storms; portable apparatus for accurately recording the magnetic elements continuously during short periods at field-stations; observations of polar lights; the atlas of auroras; observations of earth currents; correlation between radio reception and the phenomena of terrestrial magnetism and electricity, auroras, and solar activity; relations between atmospheric pollution and atmospheric nuclei of all kinds and the atmospheric-electric elements; study of ion-counters; phenomena of lightning; terminology; methods of applying the principles of geophysics to the investigation of the earth's crust; relationship with the magnetic commission of the International Meteorological Committee; observatory work; field work (magnetic surveys, provision for insuring the reoccupation at regular intervals of a sufficient number of stations all over the globe to supply necessary data regarding the secular change of the earth's magnetism); polar year 1932-1933;

secular variation of terrestrial magnetism; atmospheric conductivity; atmospheric ionization; measurement of air-earth current; observations of terrestrial magnetism and atmospheric electricity on the last cruise of the "Carnegie," by J. A. Fleming; the library of the Department of Terrestrial Magnetism of the Carnegie Institution of Washington, by H. D. Harradon.

B. Terrestrial magnetism:

Latest annual values of the magnetic elements at observatories, compiled by J. A. Fleming; investigation of the distribution of some individual changes in magnetic elements during a magnetic storm, by W. J. Peters; The geographical distribution of magnetic disturbance, by W. F. Wallis; ordinate-change integrator by C. Huff; preliminary note on a measure of magnetic activity using an ordinate integrator, by C. R. Duvall; isopors and isoporic movements, by H. W. H Fisk; on the distribution of permanent repeat stations, by H. W. Fisk and J. A. Fleming; on the earth magnetic observations on the ice, by J. Keränen; a portable magnetometer of the small type (Ordnance Survey); magnetism of rocks; observations of sudden variation of terrestrial magnetism in 1928 and 1929, by S. Ono; Disturbance in the magnetic observations caused by electric works, by S. Ono; Analysis of periodical variations; components of the diurnal amplitude of declination, by H. Labrouste; Memorandum on an international collaboration for the advancement of studies of the influence of the moon on geophysical phenomena, by D. la Cour and S. Chapman; on the theory of the diurnal solar and lunar variations of the terrestrial magnetic field, and of the magnetic perturbation, by S. Chapman; the measurement of potential gradient in the upper atmosphere, by F. J. W. Whipple; on the properties of the upper atmosphere, by B. Rolf; on the temperature of the upper atmosphere, by G. M. B. Dobson.

C. Atmospheric electricity:

The diurnal variation of the electric potential of the atmosphere over the oceans, by W. C. Parkinson and O. W. Torreson; the importance of atmospheric-electric observations at sea, by O. H. Gish; note on the establishment of observatories for the measurement of atmospheric electricity, by G. R. Wait; some general information which should be included in reports of atmospheric-electric observations, by O. H. Gish; Aitken condensation-nuclei and atmospheric potential-gradient at Washington, by G. R. Wait; note on investigation of electrical conditions in the upper atmosphere, by M. A. Tuve; the use of resistivity measurements in the detection of mineralized areas, by W. J. Rooney; the measurement of potential gradient in the upper atmosphere, by F. J. W. Whipple; studies on the electric field of the atmosphere, by Ch. Maurain; the periodic variations of atmospheric potential gradient, by L. Rodes; on some measurements of atmospheric electricity on the Mont Blanc and in Sahara, by Jean Lugeon; electrical conductivity of the air and of the atmospheric dusts on the Vesuvius, by Francesco Signore; on the particles in suspension in the atmosphere, by Denise Daude; on the atmospheric ionization, by Ch. Maurain and E. Salles; the measurement of air-earth current, suggestion from F. J. W. Whipple; phenomenon of the lightning, by M. Mathias; fulminating matter, by M. Mathias.

D. Aurora borealis:

Memorandum on several recent investigations concerning aurora borealis and phenomena connected with it, by Carl Störmer; letters and notes answering Störmer's inquiry on aurora borealis; proposition for researches relative to the synoptical distribution of the aurora borealis, by D. la Cour; on the desirability of a very exact determination of the wave length of the line 5777 Å from the aurora display, by Helge-Petersen; on the measurement of the length of the wave of the main rays of the aurora borealis, by L. Vegard.

Part V. Contains questions discussed by the section of terrestrial magnetism and electricity together with other sections, and the last.

Part VI. Gives some resolutions and wishes expressed by the section.

A list of addresses of the members who took part in the meeting is added.

A brief review of the proceedings of the section of electricity and magnetism is given by J. A. Fleming in the Journal of the Washington Academy of Sciences, vol. 21, No. 5, March 4, 1931, pp. 90-92. It reads as follows:

> Six scheduled and well-attended meetings of the Association of Terrestrial Magnetism and Electricity at Stockholm were held during August, 1930, and the Association also took part in four joint sessions - one with the Associations of Geodesy and Seismology, one with the Association of Meteorology, and two with the International Scientific Radio Union. The agenda for the meeting held an unusually great number of items, all of which were actively reported upon and discussed despite the comparatively short time available.
>
> Detailed progress reports of magnetic-survey work and investigations were received from twenty countries, including three reports from the United States -- the Coast and Geodetic Survey, the American Geophysical Union, and the Department of Terrestrial Magnetism of the Carnegie Institution of Washington. Numerous publications relating to these reports were distributed, and one was impressed not only with the amount of useful work in the fields of the Association being done throughout the world but also with the vast amount still to be done.
>
> The report of the special committee on the preparation of a photographic atlas of aurora with type descriptions and instructions for photographic and visual observation was received and approved, and authority was given that copies of the atlas be distributed without charge to observatories and organizations where worth-while auroral observations might be made. Following the report of the special committee on criteria of measures of magnetic activity, the formulas for characterization of days $(HR_H + ZR_Z)$ / 10,000 or $(NR_N + WR_W + ZR_Z)/10,000$ were adopted, where R represents the absolute daily range of the element indicated for the

Greenwich day, and arrangements were proposed that data derived by one or the other formula be published by the International Commission of Terrestrial Magnetism and Electricity in its regular publication of magnetic character of days.

The importance of continuing comparisons of standard electromagnetic instruments of various governments was emphasized, as also the design of portable apparatus of this character. It was agreed that the various suggestions on the subject of a uniform terminology be published for further consideration. The adoption of Greenwich mean time for the publication of magnetic data was referred for further consideration to a special committee. A reporter was appointed to summarize the progress in the studies of theories of terrestrial magnetism.

The importance of standardizing ion counters was emphasized, and the Department of Terrestrial Magnetism of the Carnegie Institution of Washington was appointed as a central office to which matters could be referred pertaining to ion counters for compilation, discussion, and determination of the standards. Following discussion of the electric field of the atmosphere, it was agreed that tabulations to determine electric character of the day might best be limited to electrically calm days.

Among the communications of particular interest in atmospheric electricity was one on the direct recording of air-earth current at the Kew Observatory by the C.T.R. Wilson method; such records may be compared with indirect determination through the records also being made of the conductivity and potential gradient of the atmosphere.

Considerable attention was given economic aspects of the Association's activities. These included the application of geophysical principles to the investigation of the earth's crust; it was agreed that a committee be appointed jointly by the associations of Seismology, Geodesy, and Terrestrial Magnetism and Electricity to collaborate in the solution of problems in geology through geophysics. Another economic aspect was the development of machine methods to facilitate the complex computations and compilations required in studying the numerous data accumulated by many observatories. An example of this was brought out in the report of a special committee on international collaboration for the advancement of studies of the influences of the moon on geophysical phenomena developing a plan for the assembling of data making use of Hollerith methods for compilations. The report was favorably considered and satisfaction expressed that a practical trial of the application of such a method was in prospect, thus paving the way for a definite proposal for international cooperation later.

Much attention was also given the proposal for the Jubilee Polar Year of 1932-33 of the Polar Commission of the International Meteorological Committee, and the desirability of adherence in the undertaking of all governments was stressed. Following a joint meeting of the Associations of Meteorology and of Terrestrial Magnetism and Electricity and the deliberations of a special joint committee, the following resolution was prepared for and later adopted by the General Assembly:

"The Union accepts the invitation of the International Meteorological Committee to cooperate in organizing and carrying out a second Polar Year with a similar object to that of the first Polar Year 1882-1883, and appoints the following Commission for this purpose: Störmer (Chairman), Chapman, La Cour, Maurain, and Wehrle."

Other resolutions proposed by the joint committee were adopted by the Association. One emphasizes the very great importance for the advancement of geophysical science for the Polar Year as planned and its approval that the observations should not be confined only to polar regions. The Association, realizing the desirability that all cameras, plates, and spectroscopes used in the observations of the aurora should be of equal sensitivity, voted 15,000 gold francs for the provision of instruments of a standard type. It was further unanimously resolved that all observations should be reduced according to an agreed plan and that the Commission for the Polar Year should consider the best method for making the detailed results available for all those interested, further suggesting that all published volumes should be put on sale and that the various associations of the Union should subscribe for a number of copies.

Regarding possible overlapping of the work being done by the Association of Terrestrial Magnetism and Electricity and the Commission on Terrestrial Magnetism and Electricity of the International Meteorological Committee, it was unanimously agreed upon that it is not necessary to set a rigorous definition of the domain of each organization, as no difficulties have been met with in practice and no unnecessary duplications have been encountered, and as the respective officers can continue their effective cooperation in avoiding these.

The importance of study of the correlation of the reception of wireless signals and geophysical phenomena was referred to two joint meetings with the International Scientific Radio Union, the Association expressing itself as approving any program assuring the broadcasting of cosmic phenomena to facilitate the study of correlations concerning radio communication and the magnetic and electric condition of the Earth.

Recognizing the vital need of a better world-wide distribution of observatories, especially in the southern hemisphere, a special committee was appointed to consider existing and desirable distribution of magnetic and electric observatories and to consider plans for better coordination of work and publications of existing observatories. The economic impossibility of realizing more than a limited number of observatories and the reports on secular-variation investigations submitted stressed the need of systematic field work, and a special committee was appointed to plan and to accomplish means to secure through cooperation of interested governmental and private organizations well-distributed secular-variation data. In connection with this subject, appreciation of and generous comment was expressed on all sides on the magnetic and electric work at sea secured by the "Carnegie" together with expressions of regret that the work that vessel and her commander and staff had so well done could not have been continued as planned. -

8. GEOLOGY

(447) MICROSCOPIC SUBSURFACE WORK IN OIL FIELDS OF THE UNITED STATES

By R. D. Reed

Bulletin of the American Association of Petroleum Geologists, Houston, Tex., vol. 15, No. 7, 1931, pp. 731-754.

On the basis of information furnished by subsurface workers in different districts, an attempt is made in this paper to describe the present status of the applications of the microscope to the solution of subsurface problems in the oil fields of the United States. Corresponding with the geology of the different areas, the development of microscopic work has assumed widely different forms, but the degree of success attained seems to be uniformly high. Contributions to regional geology made by the new methods are already great, and the promise for the future is bright.--Author's abstract.

9. NEW BOOKS

(448) Angenheister, G. Geophysik. II Teil. Physik des festen Erdkörpers und des Meeres (Geophysics, Part II, Physics of the solid part of the earth and of the ocean). Unter der Redaktion von G. Angenheister bearbeitet von A. Defant, F. Hopfner, K. Jung, G. Kirsch, F. Kossmat, G. Krumbach, O. Meisser, H. Schmehl, G. Tammann, E. Tams. Wien-Harms, Handbuch der Experimentalphysik, Band 25, II Teil, Leipzig, Akademische Verlagsgesellschaft m.b.H., 1931, xiii + 823 pp., 313 figs., 3 tables. Price, 74 marks, bound 76 marks. The present volume is the second of three parts which constitute volume 25 (Geophysics) of Wien-Harms monumental Handbuch der Experimentalphysik. A brief review of the book is given by H. D. Harradon in Terrestrial Magnetism and Atmospheric Electricity, Baltimore, Md., vol. 36, No. 2, 1931, p. 147.

10. PATENTS

(449) METHOD OF EXPLORING THE SUBSOIL

Richard Ambronn, of Göttingen, Germany.

United States patent 1,805,900 Patented May 19, 1931.

This patent discloses a method of exploration of the subsoil comprising feeding it with alternating current under control of the form of oscillation ellipses of the field produced thereby and measuring the field of the ground currents. For this purpose at least two current systems of like frequency and with phase displaced with respect to each other are excited in the ground. By suitable correlation of such current systems within the field of investigation it is possible to make the polarized electrical current field produced thereby approximately circular.

Claims allowed - 5.

(450) METHOD OF AND APPARATUS FOR LOCATING DEPOSITS OF OIL, GAS AND OTHER DIELECTRIC SUBTERRANEAN BODIES

Louis C. Billote, of Revere, and Edward Lipson, of Chelsea, Mass., assignors to Oil Finding Corporation, a corporation of Delaware.

United States patent 1,808,397 Patented June 2, 1931.

This invention relates to an improved method and apparatus for uses in the location of deposits of oil and gas; it may also be used in locating other subterranean deposits having relatively high dielectric characteristics compared with the surrounding medium.

The invention is based on the fact that when a current of relatively hig frequency is permitted to enter the earth through spaced electrodes, the observed effect is different when a deposit of a high dielectric nature, such as oil or gas, lies beneath the electrodes than when such deposit is not present.

Claims allowed - 14.

(451) METHOD AND APPARATUS FOR LOCATING UNKNOWN CONDUCTIVE BODIES

John J. Jakosky, of Los Angeles, Calif., assignor to the Radiore Co., of Los Angeles, Calif., a corporation of California.

United States patent 1,811,547 Patented June 23, 1931.

This invention relates to the location of orebodies, pipe lines or other bodies of relatively good electrical conductivity, within the earth's crust or any mass of relatively poor conductivity, and particularly to the location of

such bodies by methods known as "inductive methods." The method comprises energizing the conductive body by means of an electromagnetic field transmitted from an energizing loop, and then, by means of a direction-finding coil, detecting the presence and measuring the effect of the secondary electromagnetic field resulting from the induced current in the conductive body at a plurality of points adjacent to the energizing loop, the direction-finding coil being rotated, for the purpose of making such measurements, about an axis substantially parallel to the direction from the coil to the energizing loop.

Claims allowed - 8.

(452) METHOD OF DETERMINING THE STRAIGHTNESS OF DRILL HOLES IN THE EARTH

Esme Eugene Rosaire, of Fort Worth, Texas, assignor to Geophysical Research Corporation, of New York, N. Y., a corporation of New Jersey.

United States patent 1,811,648. Patented June 23, 1931.

The method of determining the deviation from the vertical of a bore formed in the earth comprises producing sound waves at different points on the earth's surface and recording the time intervals required by the sound waves to travel from each of the points to a selected place in the bore below the surface of the earth and calculating the position of the points from recorded time intervals, the known distance of surface points from the mouth of the bore, and the known distance of the selected points from the mouth of the bore.

Claims allowed - 7.

(453) METHOD OF AND APPARATUS FOR LOCATING TERRESTRIAL CONDUCTING BODIES

Theodor Zuschlag, of New York, N. Y., assignor to Swedish American Prospecting Corporation, of New York, N. Y., a corporation of New York.

United States patent 1,812,392. Patented June 30, 1931.

This invention relates to an electromagnetic device for determining the effects produced by conducting bodies, such as ore deposits, metals, water courses, salt-water beds, moist ground and the like, when an electromagnetic field is generated in their vicinity, and a method of locating and determining the proximity of the conducting bodies by the use of such a device. The apparatus comprises an exciter coil adapted to generate a primary electromagnetic field to induce a secondary field in any conducting body located in the vicinity, a plurality of receiving coils adapted to be inductively influenced by the secondary field, the receiving coils being disposed on opposite sides and in the center of the exciter coil, and providing compensating means for eliminating any inductive influence of the primary field on the receiving coils.

Claims allowed - 20.

INDEX[3]

3 - The first figure refers to the number of the abstract, the second to the method of prospecting as indicated in the Table of Contents, and the third to the page.

February, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XXXIV

BY

Frederick W. Lee

I.C.6569
February, 1932

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]

No. 34

Compiled by Frederick W. Lee[2]

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6569."

2 Senior physicist, U. S. Bureau of Mines.

1. GRAVITATIONAL METHODS

(605) DIE SCHWERE UND GEOLOGIE IN KAUKASIEN

(GRAVITY AND GEOLOGY IN THE CAUCASUS)

By P. Sawicky

Zeitschrift für Geophysik, Braunschweig, vol. 7, No. 7/8, 1931, pp. 318-323.

The existence of a relationship between geology and the course of gravity at various places must be considered an established fact at the present time. This relationship appears especially distinct if the gravity measurements are corrected for isostasy. Various gravimetric characteristics can be established for various regions in the Caucasus by using the available data for these regions. The following regions are discussed:

1. The northern Caucasus: Tertiary and post-Tertiary deposits prevail. Small positive anomalies.

2. Region of the Caucasian high mountain chain: Intrusive and crystalline rocks. Large positive anomalies.

3. Caucasia Minor: Intrusive rocks. Large positive anomalies in Armenia and in the southern part of Adjaristan.

4. The basin of the River Rion: Deposits of Paleogene period. Positive anomalies.

5. Basin of the Kura River except its lower course: Mainly post-Tertiary deposits. Small positive anomalies.

6. Black Sea shore: Positive anomalies.

7. Caspian Sea shore and the Shirvan: Tertiary and post-Tertiary deposits. Large negative anomalies.

Concerning the direct comparison of gravimetrical values with the age and densities of various formations, it is established that the values of anomalies become smaller in places where the old and dense rocks are changing into those of a younger age and into rocks less dense.

A further study of the Caucasian structure by means of pendulum methods is necessary. Best results may probably be obtained by carrying out a gravitational survey by means of profiles. An example of such a profile carried out from Tiflis to Vladikavkas is shown in a figure.--Author's abstract translated by W. Ayvazoglou.

(606) SUR QUELQUES MESURES DE LA GRAVITÉ DANS LA RÉGION PARISIENNE

(CONCERNING SOME GRAVITY MEASUREMENTS IN THE REGION OF PARIS)

By L. Éblé and E. Salles

Comptes Rendus de l'Académie des Sciences, Paris, vol. 193, No. 17, 1931, pp. 719-720.

Results of relative measurements only are given in this article. The observatory of Paris has been chosen as the station of departure. The measurements were carried out in the summer of 1931 by means of Sterneck's apparatus. Observations were made at the following places:

Observatory of Paris	- 30 observations
Parc Saint-Maur	- 13 observations
Beauvais	- 12 observations
Versailles	- 11 observations

The results of observations and the values of g obtained are shown in a table.--W. Ayvazoglou.

(607) PROSPECTING FOR NEW OIL-BEARING DEPOSITS MUST BE EXPEDITED (IN RUSSIAN)

By B. Numerov

Vestnik of the Geological and Prospecting Service of U.S.S.R., Leningrad, vol. 5, No. 5-6, 1930, pp. 7-9

The necessity of a general magnetic and gravimetrical survey in order to establish the existence of deep tectonic disturbances by which the probability of anticline structures in the upper oil-bearing formations may be expected is emphasized.

Good results in using the gravimetrical method of prospecting in the oil-bearing regions of Emba and Grozny for disclosing salt domes are mentioned.

The importance of the aerial survey in plotting the data of geological observations on the map, and the low cost and quickness of such a survey (about $1 per square kilometer and about 1,000 square kilometers per day) are, according to Numerov, factors which absolutely must be taken into consideration in prospecting for new oil fields.

In addition to prospecting for oil-bearing regions, the gravitational survey had proved to be of great value also in Krivoy Rog and the Don Basin.--W. Ayvazoglou.

(608) ÜBER DIE GENAUIGKEIT DER MESSUNG MIT DER DREHWAAGE

(ON THE ACCURACY OF TORSION BALANCE MEASUREMENTS)

By Richard Schumann

Zeitschrift für Instrumentenkunde, Berlin, vol. 51, No. 8, 1931, pp. 426-430.

In this article the author gives mathematical consideration to the remainders - that is, the difference between the values observed and those obtained by calculation - in the representation of the observations, if measurements were taken by observing at more than 5 azimuths.--W. Ayvazoglou.

(609) GENERAL CHARACTERISTICS OF THE GRAVITY METHOD OF PROSPECTING ACCORDING TO THE FIELD WORK PERFORMED BY THE FORMER GEOLOGICAL COMMITTEE IN 1925 - 1928

By B. Numerov

Transactions of the Geological and Prospecting Service of the U.S.S.R., Leningrad, No. 36, 1931, 154 pp.

Numerov sums up the work done by the Geological Committee since 1924 when he was sent to Germany to get acquainted with the results and organization of the work being done there with the gravitational variometer.

The first part of the book deals with the methodical work and the second with the results of observations.

The articles appearing in this volume have already been printed in various publications.

Following is a list of the articles, with reference to the Geophysical Abstracts in which they were noted:

I. Methodical works:

1. Theoretical cases for the use of gravitational methods in geology, by Numerov (Geophys. Abs. 1).

2. Fundamental formulas for the analysis of torsion balance observations, by Numerov (Geophys. Abs. 31).

3. Normal effect of the earth's ellipsoid upon the derivatives of the gravity potential, by Numerov (Geophys. Abs. 31).

4. Corrections of observations made by means of a torsion balance with respect to topography, by Numerov (Geophys. Abs. 5).

5. Analytical method of calculating the influence of topographic masses, by Numerov (Geophys. Abs. 31).

6. Graphical method of calculating the influence of topographic masses upon the observations made by a torsion balance, by Samsonov (Geophys. Abs. 31).

7. Influence of the extraneous masses upon gravity observations in case of their infinite extension, by Numerov (Geophys. Abs. 31).

II. Results of observations:

1. Results of gravimetric observations on the Shuvalovo Lake in the winter of 1927 and 1928, by Numerov (Geophys. Abs. 10).

2. Gravity observations in the Solikamsk and Berezniaky district in the Northern Urals in 1926 and 1927, by Numerov (Geophys. Abs. 10).

3. Results of the general gravity survey in the Emba district, by Numerov (Geophys. Abs. 10).

4. Results of gravity observations in Krivoy Rog in 1928, by Alexandrov (Geophys. Abs. 31).

5. Results of gravitational observations in the region of Grozny in 1928, by Numerov (Geophys. Abs. 10).

6. Results of gravity observations of 1928 near the Baskunchak Lake, by Numerov (Geophys. Abs. 10).

The book may be obtained from the Geological and Prospecting Service of the U.S.S.R., Leningrad -- 26 Vassily Ostrov, Sredny Prospect No. 72-b. Price, Rouble: 3.50.--W. Ayvazoglou.

2. MAGNETIC METHODS

(610) DR. FILCHNER'S ERDMAGNETISCHE BEOBACHTUNGEN IN ZENTRALASIEN 1926 BIS 1928. BEARBEITET VON O. VENSKE

(DR. FILCHNER'S EARTH-MAGNETIC OBSERVATIONS IN CENTRAL ASIA FROM 1926 TO 1928. REVISED BY O. VENSKE)

By K. Haussmann

Zeitschrift für Geophysik, Brandenburg, vol. 7, No. 7/8, 1931, pp. 355-359.

Haussmann gives a brief description of the earth-magnetic observations carried out by Dr. Filchner from 1926 to 1928 during his 6,000 kilometer journey through China and Tibet. The whole material has been revised by O. Venske. Earth-magnetic elements D, H, and I were measured at 157 stations.

According to the investigations made by Venske concerning the errors in measurements the following results were established: Errors in D were very small and not over 1'; the azimuth error was about 5', thus the mean error of the D value was 6'. The mean error of the horizontal intensity reached 15 γ. The mean error of the inclination values was 4'.

Filchner expects to continue his observations in Central Asia as soon as possible.--W. Ayvazoglou.

(611) ON THE MYSOVSKI IRON-ORE DEPOSIT IN THE BURIAT-MONGOLIAN AUTONOMOUS SOCIALIST SOVIET REPUBLIC (IN RUSSIAN)

By P. J. Kasatkin and S. S. Smirnov

Bulletin of the Geological and Prospecting Service of U.S.S.R., Leningrad, vol. 50, No. 29, 1931, pp. 457-489.

The Mysovski iron-ore deposit is situated between the Levaia (left) Mysovaia River and Pravaia (right) Mysovaia River, about 14 kilometers to the south of Lake Baikal and of the Mysovaia station of the Transbaikalian railway line.

Favorable conclusions regarding this deposit drawn from investigations made since 1897 caused the Central Geological Committee to organize, during the summer of 1928, a magnetometric survey of this region.

The lines of prospecting were chosen at intervals of 20 meters and the observation points along them at intervals of 1 to 10 meters. Tiberg-Thalen's magnetometer and a universal deflectory magnetometer (model of the former Geological Committee) were used.

According to various geological investigations the deposit was supposed to be a "magnetite lode of the type of a bed vein." By the magnetic survey it was established that the orebody consisted of a series of separate ore lenses. Based on the interpretation of the data obtained by the survey, it was also established that the separate deposits have an insignificant thickness relative to their horizontal extension. By taking the figures into account for the evaluation of the ore reserves a conclusion was drawn that the deposits can not be considered as having industrial value.--W. Ayvazoglou.

(612) CONCERNING CALCULATIONS OF COSTS OF GEOPHYSICAL PROSPECTING (IN RUSSIAN)

By A. Strona

Vestnik of the Geological and Prospecting Service of U.S.S.R., Leningrad, vol. 5, No. 7-8, 1930, pp. 18-19.

In this article Strona emphasizes the necessity of keeping records of the costs involved in various methods of geophysical prospecting, as well as of the efficiency of the work resulting from using various kinds of instruments.

A comparison between the work with the deflector-magnetometer constructed by the Geological Committee of the U.S.S.R. and the Tiberg-Thalen magnetometer is given as an example. According to the records from a magnetic survey carried out in 1929 in the region of Krivoy Rog, 976 points were determined during 42 days (23 points per day) by the Geological Committee's magnetometer, and 17,527 points during 172 days (102 points per day) by the Tiberg-Thalen magnetometer. Thus the costs of the survey may be greatly reduced by working with the latter magnetometer.

Detailed records are especially important in working on large scale.

Analyzing the results of magnetic surveys carried out during the years 1927-1930 with the Tiberg-Thalen magnetometer in the region of Krivoy Rog the mean cost for one point of measurement was found to be about 25 cents, depending on the thickness of the network.--W. Ayvazoglou.

3. SEISMIC METHODS

(613) SEISMISCHE BEOBACHTUNGEN BEI STEINBRUCHSPRENGUNGEN

(SEISMIC OBSERVATIONS MADE IN CONNECTION WITH BLASTING IN QUARRIES)

By Bernhard Brockamp

Zeitschrift für Geophysik, Braunschweig, vol. 7, No. 7/8, 1931, pp. 295-317.

Seismic records on blasting in quarries obtained by the Geophysical Institute of Göttingen to 1929 were revised and completed by four new observations made during 1929.

The travel-times were reduced to a common sea level. The influence of the geologic difference of the soil below the stations upon the travel-time was taken into consideration. Notwithstanding this a deviation from the mean time-curve was observed for single stations; this allowed the author to draw conclusions on the local thickness of the overburden of the rocks. The P_2 layer situated below the overburden was determined to be varicose primitive rocks of a thickness of 6 to 7 kilometers. The lower border of the P_2 layer sinks to the south of Göttingen down to 12 kilometers and rises to $7\frac{1}{2}$ kilometers in the southwest and southeast directions.

The velocity of P_2 waves was 5.9 kilometers per second, and of P_3 waves 6.7 kilometers per second. P corresponds to $\overline{P}$, and P_3 to P^* of nearby earthquakes.--Author's abstract translated by W. Ayvazoglou.

(614) PRINCIPLES UNDERLYING THE INTERPRETATION OF SEISMOGRAMS

By Frank Neumann

Coast and Geodetic Survey, Washington, D. C., 1931, 31 pp., 5 plates.

Contents:

1. Some important operation problems (intensity of light spots; focussing; time control; atlas of seismograms).

2. Nature of earthquake vibrations.

3. Elementary wave types, P, S, L.

4. Dependent wave types.

5. Response of seismograph to earthquake vibrations (magnification equation; harmonic magnification illustrated; acceleration equation; effect of damping).

6. Travel-time charts and tables (determination of distance and time of origin from seismological table; graphical determination of distance and time of origin; difficult interpretations; comparison of travel-time tables; travel time at short distances).

7. Interpretation of seismogram and preparation of report (artificial disturbances--tilting, short period vibrations, wind tremors, convection currents; microseisms; definition of "activity;" precision required in reading times of activities; interpretation of activities when epicenter data are unknown; interpretation of activities when epicenter data are known).

8. Computation of ground displacement and acceleration.

9. Determination of direction of ground movement and azimuth of epicenter.

10. Descriptive illustrations and plates (paths of seismic waves; motion of earth particle in two types of surface wave; nomenclature; travel-time chart; magnification curves; seismograms). .

The notes published in this pamphlet are intended to serve as a guide to the seismologist whose task is to supervise the operation of a seismographic station.--W. Ayvazoglou.

(615) COMPARATIVE STUDIES OF EARTHQUAKE MOTIONS ABOVE GROUND AND IN A TUNNEL (PART I)

By Nobuji Nasu

Bulletin of the Earthquake Research Institute, Tokyo, vol. 9, No. 4, 1931, pp. 454-472.

Comparative studies of earthquake motions carried out inside of the Tanna tunnel and above it with a modern seismograph manufactured from stainless steel are described. According to the results of investigations the earthquake motions inside of the tunnel were found to be generally smaller than those on the surface of the ground. The amplitude inside of the tunnel was taken as the unit for comparing that above the ground.

The relation between the ratio of earthquake motions above the ground to that in the tunnel and the period of earthquake are shown in a diagram.

Further studies with modern seismographs for obtaining more data are desirable.

Tables, diagrams, and seismograms complete the article.--W. Ayvazoglou.

(616) CARACTÉRISTIQUES DES ONDES SÉISMIQUES D' APRÈS LES ENREGISTREMENTS ACCÉLÉROMÉTRIQUES

(CHARACTERISTICS OF SEISMIC WAVES ACCORDING TO THE REGISTRATION OF ACCELERATIONS)

By Mishio Ishimoto

Bulletin of the Earthquake Research Institute, Tokyo, vol. 9, No. 4, 1931, pp. 473-484.

Figures showing the acceleration registration of seismic shocks obtained in the basement of the Earthquake Research Institute in Tokyo and a table revealing the time of their occurrence and the places of epicenters are given. Their general characteristics are discussed.

The following two chapters deal with the periods of the shocks and with their structure.--W. Ayvazoglou.

(617) SEISMOMETRICAL REPORT

By N. Nasu and Ch. Yasuda

Bulletin of the Earthquake Research Institute, Tokyo, vol. 9, No. 4, 1931, pp. 509-513.

The report includes data of perceptible earthquakes originated within a distance of 160 kilometers from Tokyo and felt there, for the period April 1 to June 30, 1931.

For previous report see Geophysical Abstracts No. 32 of December, 1931.--W. Ayvazoglou.

(618) THE INSTRUMENTAL PHASE-DIFFERENCE OF SEISMOGRAPH RECORDS; AN ILLUSTRATION OF THE PROPERTIES OF DAMPED OSCILLATORY SYSTEMS

By F. J. Scrase

Proceedings of the Physical Society, Cambridge, vol. 43, Part 3, No. 238, 1931, pp. 259-273.

A discussion is given of the method of interpretation of the maxima shown on the records of earthquakes during the surface-wave phase. The usual procedure is to treat the waves (which actually appear as beats) as being truly simply harmonic and to apply the formulas which are derived on this assumption. It is shown that, in general, this procedure does not necessarily lead to the correct interpretation. In the case of direct registration the true earth maximum may have occurred one half-period later than the time obtained by the usual correction. With galvanometric registration the maximum may have occurred either one, two, or three half-periods earlier than the time indicated by the usual formula of Galitzin. Some curves are included to illustrate these points, and an attempt is made to obtain a mathematical explanation.

It is shown that there is no easy method of eliminating an ambiguity of one half-period. For direct registration, therefore, the phase-correction at present in use appears to be as good as the one alternative. In the case of galvanometric registration, although there are altogether four forms of phase-correction, the number of alternatives for any particular period can not exceed two. The final recommendation in this case is that the correction suggested by Somville and which is one half-period less than Galitzin's, be adopted for general use.--Author's abstract.

(619) GEOPHYSICAL PROSPECTING WITH EXPLOSIVES AND THE SEISMOGRAPH

Atlas Powder Co., Wilmington, Del., 1931, 40 pp.

This pamphlet has been prepared by the Atlas Powder Co. in collaboration with Dr. J. Brian Eby for the purpose of disseminating information on the

history and the background of this new method of exploring for minerals, describing elementary aspects of the theory involved in the use of the seismograph, and providing details on explosives adaptable for this work.

Contents of the book:

1. Foreword.

2. Geophysical prospecting (general idea is given).

3. Historical sketch (development of instruments and theories; description of the mechanical seismograph and electrical seismograph in most common use; aspects of the refraction method and reflection method of geophysical prospecting).

4. Application of seismic methods.

5. Explosives as applied to geophysical prospecting (character of explosives used; their characteristics and properties).

6. Comparison of powders.

7. Methods of planting the charge.

8. Blasting caps and electric method of blasting.

9. Experimental data (seismic velocities in various formations; relation of depth to charge).

10. Future of explosives in geophysical prospecting.

11. Explosives for seismographic work abroad.

12. Conclusions.

13. Glossary of terms used.

Tables and figures are added.-- W. Ayvazoglou.

(620) INVESTIGATION ON THE DEFORMATION OF THE EARTH'S CRUST IN THE TANGO DISTRICT CONNECTED WITH THE TANGO EARTHQUAKE OF 1927

By Chuji Tsuboi

Bulletin of the Earthquake Research Institute, Tokyo, vol. 9, No. 4, 1931, pp. 423-434.

This is a continuation of an article published by the author in the Bulletin of the Earthquake Research Institute, Tokyo, vol. 8, No. 2, 1930, pp. 153-222 (see Geophys. Abs. 18, October, 1930).

The purpose of the present paper is to discuss the results of the fith survey carried out with the support of the Imperial Academy and the Land Survey Department. The levelling route covered by the present survey is shown in a map. The results were compared with those obtained from the fourth post-seismic levelling, and the changes of heights of benchemasses in the Tango district are given in a table.--W. Ayvazoglou.

4. ELECTRICAL METHODS

(621) TOPOGRAPHICAL STUDY OF A HIDDEN BEDROCK SURFACE BY RESISTIVITY MEASUREMENTS

By E. G. Leonardon

The Engineering Journal, Montreal, vol. 14, No. 6, 1931, pp. 331-335.

Before describing the field survey Leonardon discusses briefly the elementary principles underlying the measurements of the specific resistance of large volumes of the ground. The author's summary reads as follows:

> This article describes the results of an electrical exploration carried out in 1929 for the Department of Railways and Canals, in connection with the study of a dam site at Morrisburg, Ont., to obtain further data on the underground contours of the bedrock.
>
> The method adopted was developed by Professor Schlumberger, and depends on the measurement of an electrical field set up in the ground by current passed between two electrodes at the ends of a known base line. Several series of measurements are taken with different lengths and directions of base line, thus determining many equipotential curves from which the average resistivity of the subsoil can be deduced. Most rocks have resistivities of 1,000 to 2,000 ohms per meter-meter square, while clayey formations give corresponding figures of 10 to 400 ohms.
>
> The necessary apparatus is described, and results given for a large series of measurements carried out on both sides of the St. Lawrence River at Morrisburg. Good agreement was obtained between the depths calculated by the electrical method and those measured by actual drilling.

In the conclusion to this article the author shows that the electrical method of prospecting is rapid, economical, and especially applicable at the beginning of a preliminary survey, at which time heavy expenditure can be avoided by careful planning to limit, as far as possible, exploration by drill holes, test pits, and other expensive means.

The work described in this article was performed in 30 working days. During this time 101 depth determinations were made at a cost of less than $4,500.

Five illustrations are added.--W. Ayvazoglou.

(622) EINE ELEKTRODYNAMISCHE METHODE ZUR ERFORSCHUNG DES ERDINNERN

(AN ELECTRODYNAMIC METHOD FOR THE INVESTIGATION OF THE INTERIOR OF THE EARTH)

By H. Löwy

Physikalische Zeitschrift, Leipzig, vol. 32, No. 8, 1931, pp. 337-345.

The article deals with the question of the physical constitution of oil. Experiments carried out by the author in southern California in 1928 are described.

After a brief discussion on the question of whether the soil can be investigated by electrical means in arid regions, the author gives the results of his examinations in various places in California, based on which he draws the conclusion that the oil-theory established by him must be considered to be correct.--W. Ayvazoglou.

(623) SOME EXPERIMENTS RELATING TO GEOPHYSICAL PROSPECTING

By D. C. Gall

Journal of Scientific Instruments, London, vol. 8, No. 10, 1931, pp. 305-313.

The article concerns the Equiquadrature method recently developed by the author. In an introduction to this article, written by A. Broughton Edge, the latter, referring to some difficulties arising in connection with the equipotential line method, mentions that in order to solve these difficulties he proposed to Gall to work out an entirely new problem - that of locating equipotential lines for the out-of-phase field. A method for plotting these lines directly upon the ground was then devised by Gall and it was agreed to call these lines the "Equiquadrature or EQ lines."

In this equiquadrature method two systems of equipotential lines are plotted. The two systems are in time-quadratures, and are called EP and EQ lines. The EP lines are almost identical with the equipotential lines of the older method, but in plotting them the spacing is chosen in known increments of potential. The EQ lines are 90° out of phase with the EP lines, and these also are plotted at known increments of potential. These two line systems represent the entire surface potential phenomena of elliptical polarization in its simplest form. The configuration of the line systems supplement each other in the indication of anomalies.

Experiments made on a model in order to become familiar with typical configurations are described.

The scheme is shown in a figure.--W. Ayvazoglou.

(624) ELECTRICAL SURVEY OF LIMESTONES OF THE BIBI-EIBAT BAY

By V. Melikian

Azerbaijanskoye Neftianoe Khoziaistvo, Baku, vol. 11, No. 9-10, 1931, pp. 104-108.

Melikian's article deals with the application of an electrical submarine survey for establishing geological structures. This work has been carried out in the U.S.S.R. for the first time and, according to the author, may be of great importance in prospecting for oil below the Caspian Sea.

Practical operation of the method is described and illustrated by figures. Difficulties consisted mainly of damages to cables caused by sharp stones. Strong winds and rough sea often reduced the efficiency of the work, as it was difficult to place the cables in the direction required.

The results of the electric measurements are given in a map showing the profiles and lines of equal resistance.

Geological results of the survey are described, and the possibility of the submarine survey of the littoral of the peninsula of Apsheron is discussed.--W. Ayvazoglou.

(625) ELECTRICAL PROSPECTING FOR GOLD VEINS

By Hans Lundberg

The Northern Miner, Toronto, vol. 17, No. 46, January, 1932, p. 4.

A case of the application of a method by using the apparatus known as the "Racom" for the survey of the property of the Metals Development, Ltd., near Woman Lake in western Ontario is described. The work was carried out by running Racom profiles across the supposed strike of the veins and at a depth sufficient to cut below the overburden, which as a rule did not exceed 30 feet.

Other examples of prospecting for quartz veins in Sumatra and of tracing quartz veins in the Philippine Islands in an area where gold mining had been carried on during ancient times are mentioned.

The author concludes that this method has recently been further developed into what is known as the Differential Rate or Change Method; this consists largely of a new way of interpreting the field results, whereby the influence of the overburden is eliminated.--W. Ayvazoglou.

(626) BEMERKUNGEN ZUM GEOELEKTRISCHEN NACHWEIS UND ZUR MIGRATION DES ÖLS

(REMARKS ON THE GEOELECTRIC INDICATION AND ON THE MIGRATION OF OIL)

By J. Koenigsberger

Petroleum Zeitschrift, Berlin, vol. 27, No. 32, 1931, p. 579.

In a brief discussion of the migration of oil the author remarks that this migration, which is not concluded yet, may also be indicated by geoelectric observations. Bad conductors in layers of normal conductivity may be well observed geoelectrically. This was proved by results of experiments made by many scientists and it was recently shown (by Lee and Swartz, U. S. Bureau of Mines, Technical Paper 488) that oil deposits lying close to the surface could be determined according to Wenner's four points method. An error is often made in that geoelectric effects are attributed to productive oil deposits at a depth of from 800 to 1,000 meters. Theoretical calculations show that the deviation of the current can not be caused by layers at depths below 500 meters.

Therefore these deviations are certainly caused by intermediate stations of the oil migration situated above the productive oil deposits.

Some remarks concerning Koenigsberger's viewpoint on the factors influencing the conductivity of sedimentary rocks in oil deposits are made by H. Hedström. Hedström's remarks are answered by Koenigsberger (see Petroleum Zeitschrift, vol. 27, No. 40, 1931, pp. 731-732).--W. Ayvazoglou.

5. RADIOACTIVE METHODS

(627) THE IONIZATION OF THE ATMOSPHERE MEASURED FROM FLYING AIRCRAFT

By D. C. Rose

Canadian Journal of Research, Ottawa, vol. 5, No. 6, 1931, pp. 625-635.

The present paper deals with an attempt to study the variation in the ion content of the air at different altitudes. Author's abstract reads as follows:

> The Gerdian type of atmospheric ionization measuring apparatus was attached to a cabin aeroplane so that the state of ionization of the atmosphere could be studied. The limitations of the apparatus for aeroplane use are discussed. Measurements were taken from ground level to heights of 15,000 feet. The results are plotted in number of ions per cubic centimeter (separate curves for positive and negative) at different altitudes.

The results indicate that at the cloud level there is an abnormal excess of small positive ions and a minimum in the excess of positive ions over negative ions from 4,000 to 6,000 feet higher. This does not include large ions such as charged water drops or dust particles. The observations were taken in regions free from clouds, the cloud level being determined by observations on clouds in the sky, and by relative humidity measurements taken at the same time.

In view of the preliminary nature of the results described, the conclusions drawn must be, according to the author, considered only as tentative and more observations should be made with the potential gradient apparatus as well as ionization measuring equipment.--W. Ayvazoglou.

(628) DETERMINATION OF THE RADIOACTIVITY OF NATURAL WATERS AND SOME RESULTS FOR FLOWING ARTESIAN WELLS

By James A. Hootman

American Journal of Science, New Haven, vol. 22, No. 131, 1931, pp. 453-463.

Based on the fact that traces of radium are to be found in all kinds of ordinary rocks, the author draws the conclusion that nearly all natural springs and well waters will contain electroscopically detectable traces of dissolved radioactive materials, notably the rare inert gases, radium emanation and thorium emanation, which are easily soluble in water. A method for such a "prospecting" for radium is discussed in this article by the author.

Apparatus is described and results of a series of field tests carried out on artesian wells in Mississippi are given.--W. Ayvazoglou.

(629) AURORES POLAIRES ET RAYONS COSMIQUES

(AURORA BOREALIS AND COSMIC RADIATION)

By A. Dauvillier

Comptes Rendus de l'Academie des Sciences, Paris, vol. 193, No. 7, 1931, pp. 348-350.

The well-known relationship between the sun spots, aurora borealis, and magnetic disturbances is of a temporary character. According to more than 100 observations made by Nordenskiöld on the lightning zone, around the magnetic pole, permanent disturbances must be expected also. They may originate from 15.10^9 volt-electrones which are connected with the penetrating radiation. Dauvillier's viewpoint on this question is discussed in the article.--W. Ayvazoglou.

7. UNCLASSIFIED METHODS

(630) GEOPHYSICAL PROSPECTING IN 1931

By Donald H. McLaughlin

Mining and Metallurgy, New York, vol. 13, No. 301, 1932, pp. 16-20.

A brief review of the progress in geophysical work during 1931 is given. The author says that although the employment of geophysical methods in the search for oil and ore has been greatly curtailed during the past year, more time has been used for perfection of instruments and field methods, for careful analysis of data, and for thorough investigation of the many problems involved in interpreting physical anomalies in terms of concealed geologic conditions; thus the achievements must be considered significant.

Notable work carried out by companies, States, and private organizations is mentioned and the results are discussed.

Barton's statement on the commercial activities and the results accomplished in the Gulf region forms one paragraph of the article.

Among the numerous publications appearing during the past years, the report of the Imperial Geophysical Experimental Survey published under the title, "The Principles and Practice of Geophysical Prospecting," and edited by A. B. Broughton Edge and T. H. Laby, and Memoir 165 of the Canadian Geological Survey entitled "Studies of Geophysical Methods in 1928 and 1929," are those deserving the highest merit. Activity of the U. S. Bureau of Mines in publishing a number of papers on current work on geophysics is mentioned.--W. Ayvazoglou.

(631) PRINCIPLES AND PRACTICE OF GEOPHYSICAL PROSPECTING

Editorial note

Queensland Government Mining Journal, Brisbane, vol. 32, No. 376, 1931, p. 366.

This is the title of the Report of the Imperial Geophysical Experimental Survey carried out in Australia and edited by A. Broughton Edge and T. H. Laby. A summary of the report is given in this article (see Geophys. Abs. 30, p. 257).
--W. Ayvazoglou.

(632) OBSERVATIONS MAGNÉTIQUES ET ÉLECTRIQUES EN SAHARA

(MAGNETIC AND ELECTRICAL OBSERVATIONS IN THE SAHARA)

By C. LeCamus and F. de Saint-Just

Comptes Rendus de l'Academie des Sciences, Paris, vol. 193, No. 15, 1931, pp. 600-601.

The authors describe some magnetic and electrical observations made during their journey in the Sahara in December, 1930, and January, 1931.

The three elements of the earth-magnetic field were measured at the following three stations: Gao, Tanesrouft and Camp Louis Marin(Hoggar). The two last stations are situated in a region where no similar measurements had been made previously. Chasselon's theodolite-compass No. 5 (small model) and Chasselon's dipping needle No. 68 (small model) were used. The results of the measurements are given in a table. The existence of a magnetic disturbance between the 20th and 25th parallels, around the 3d meridian east of Greenwich, has been proved by these observations.

Measurements of the atmospheric electric field were made at the same stations according to Moulin's method.

The following figures are given:

Tanesrouft, 1st series, 26 to 31 volts per meter (December 6, 1930);
Gao, 1st series, 18 to 32 volts per meter (December 13, 1930);
Camp Louis-Marin, 1st series, 16 to 46 volts per meter (January 4, 1931);
2nd series, 19 to 31 volts per meter (January 5, 1931);
3rd series, 24 to 41 volts per meter (January 6, 1931).
--W. Ayvazoglou.

(633) AERIAL SURVEY AND GEOGRAPHIC OBSERVATIONS AS BASES FOR THE FUTURE GEOLOGICAL AND PROSPECTING WORK (IN RUSSIAN)

By B. Numerov

Vestnik of the Geological and Prospecting Service of the U.S.S.R., Leningrad, vol. 5, No. 7-8, 1930, pp. 30-32.

The importance of the aerial survey is emphasized and the procedure for such a survey is described.

The aerial survey should be used, according to the author, in the first place for preliminary surveys carried out by magnetic and gravitational methods.--W. Ayvazoglou.

(634) DIE HAUPTVERSAMMLUNG DER DEUTSCHEN GEOLOGISCHEN GESELLSCHAFT

(GENERAL MEETING OF THE GERMAN GEOLOGICAL SOCIETY)

By Breddin

Glückauf, Essen, vol. 67, No. 48, 1931, pp. 1493-1498.

According to Breddin's report on the general meeting of the German Geological Society, which took place from the 14th to the 17th of September, 1931, the following items on geophysical methods of exploration were discussed:

1. Relative gravity measurements in oil-bearing regions. This paper was presented by Gornich, in which he emphasized the importance of torsion balance and pendulum measurements for the determination of gravity differences in the earth's crust in general, and in particular the value of the measurements for the discovery of salt domes.

2. Results of the regional-magnetic investigations in the Eifel. The report was made by Reich. The results of numerous measurements carried out by Reich in the region of the Rhine Slate mountains with Schmidt's field balance were discussed. An explanation for a magnetic maximum between Bonn and Lenz and a minimum along the Mosel in the region of Coblenz is given. Geological importance of the changes in the intensity of the magnetic field during the last 30 years, as established recently, was mentioned.

3. The results of local magnetic investigations in the volcano region of the Laach Lake were discussed by Ahrens. By the measurements carried out by him in the region of Niedermendig the existence of basalts and basaltic lavas and their extent beyond the places of previous discoveries has been proved. Very strong magnetic disturbances (up to 6,000 γ) were established above the basaltic slags of the old volcanoes in the region of the Laach Lake.--W. Ayvazoglou.

(635) GEOPHYSICAL INSTRUMENTS

Editorial note

Mining and Metallurgy, New York, vol. 13, No. 301, 1932, p. 62.

The note mentions the development made by the American Askania Corporation, Houston, of Askania magnetometers in order to eliminate the temperature effects and consequent delays in field work because of need to determine temperature coefficient of instrument at each step. According to this note the same company has also brought out a 3-pendulum apparatus for commercial as well as scientific purposes in geophysical work of special interest to oil companies.--W. Ayvazoglou.

(636) REFLEXION SENKRECHT ZU FLÄCHEN OPTISCH EINACHSIGER UND ROMBISCHER, STARK ABSORBIERENDER KRISTALLE

(REFLECTION PERPENDICULAR TO SURFACES OF OPTICALLY SINGLE-AXIAL AND RHOMBIC, STRONGLY ABSORBING CRYSTALS)

By J. Koenigsberger

Neues Jahrbuch für Mineralogie, Geologie und Paläontologie, No. 64 (A), 1931, pp. 107-121.

The author derives some theoretical conclusions on the following three methods for obtaining quantitative values concerning the microscopic investigation of the incidence of perpendicular rays of light:

1. The measurement of reflective capacity carried out by H. Frick on numerous isotropic and anisotropic minerals.

2. The measurement of phase difference of two oscillations perpendicular one to another.

3. The measurement of the anisotropy of the reflection, either directly or by means of separate measurements of the two components.--W. Ayvazoglou.

8. GEOLOGY

(637) SYMPOSIUM ON SALT DOMES

Editorial note

Journal of the Institution of Petroleum Technologists, London, vol. 17, Nos. 91 and 92, 1932, pp. 251-371.

This article contains papers presented at the 132nd and 133rd general meetings of the Institution of Petroleum Technologists held on January 13 and February 10, 1931, respectively.

It was decided that these meetings be devoted to the presentation of short summaries of the various papers and to a general discussion of the whole subject. It was also decided that discussion of a geophysical nature should be held on a future occasion as a separate symposium.

The articles presented are as follows:

1. Salt domes of North Germany, by James Romanes. Geological history; the form of the domes and their tectonic relation to the surrounding sediments; the mechanics of salt dome formation; and the influence of salt domes on the formation and concentration of petroleum were discussed.

2. Salt-dome depositional and deformational problems, by G. M. Lees. Items concerning the conditions of deposition of saline formations; salt domes of Persia; the tectonics of the Lower Fars of Persia; Palestine-Jebel Usdum; the miocene salt-gypsum series of the Gulf of Suez; the Red Sea salt domes; plasticity of salt; and the mechanism of the uprise of salt masses were reported.

3. Salt domes of Texas and Louisiana Gulf Coast, by Frederick G. Clapp. The author gives outstanding features regarding the Gulf Coast salt domes, such as: Enumeration of domes; general description; stratigraphic relationships; structural relationships; production; manner of discovery; origin; and the "cap rock."

4. Salt domes in Persia, by J. V. Harrison. Attention has been directed almost exclusively to the activities and manifestations of the older salt formation of Persia.

5. Intrusive salt bodies in Coastal Asia, southwestern Arabia, by Arthur Wade. Studies in connection with "salt dome" phenomena carried out in this region are presented by the author.

6. Origin of the salt domes of the Gulf Coastal Plain of the United States, by E. DeGolyer. A brief discussion as to the origin and age of the salt and the associated cap rock is given.

7. Moot points in salt dome theory, by Launcelot Owen. Some problems arising from the study of salt domes are investigated.

8. A contribution to "salt dome" geochemistry, by Murray Stuart. This is a contribution of the author to the knowledge of naturally occurring salt and its chemical and geological idiosyncrasies acquired by him during his work on salt formation in India.

9. Salt occurrences in Egypt, by Harold Dabell. The Egyptian coast of the Gulf of Suez, known as the oil-field region lying between latitude 28.5 and 26.46 and longitude 33.20 and 34, is considered.

The presentation of the papers mentioned above was followed by a discussion.--W. Ayvazoglou.

(638) HOBBS FIELD, LEA COUNTY, NEW MEXICO

By Ronald K. deFord and Edwin A. Wahlstrom

Bulletin of the American Association of Petroleum Geologists,
vol. 16, No. 1, 1932, pp. 51-91.

The Hobbs field is near the southwestern rim of the Llano Estacado. Structure can not be mapped at the surface; the field was discovered by geophysical surveys. The producing structure is an elongated dome. The oil and gas reservoirs are members of that Permian system which occupies the Permian basin of west Texas and southeastern New Mexico. The main reservoir is a porous, light-colored limestone about 200 feet thick. The pressure in this reservoir is more than 1,500 pounds per square inch at a depth of 4,150 to 4,200 feet. Beyond the limits of the oil pool the reservoir-rock contains water under corresponding pressure. Porosity (hence initial production) is related to structure, being greatest along the anticlinal crest, less on the flanks, least in wells off structure.

Approximately one-third of the folding occurred shortly after the deposition of the main limestone reservoir and before salt deposition. Almost two-thirds occurred after salt deposition and was probably post-Triassic, certainly pre-Pliocene.--Author's abstract.

9. NEW BOOKS

(639) Conrad, V., and Weickmann, L., Ergebnisse der kosmischen Physik (Results of cosmic physics). Gerlands Beiträge zur Geophysik, Erster Supplementband. 1931, 448 pp. with 243 figs. Akademische Verlagsgesellschaft m.b.H., Leipzig.

Contents of the book: (1) On the problems of the aurora borealis, by Carl Störmer; pp. 1-86, 70 figs. (2) The barometer effect of the penetrating radiation, by W. Kolhörster and L. Tuwim; pp. 87-126, 23 figs. (3) Absorption-coefficient of penetrating radiation, by W. Kolhörster and L. Tuwim; pp. 127-179, 20 figs. (4) The atmospheric ozone, by F. W. Paul Götz; pp. 180-235, 41 figs. (5) On the propagation of the explosion waves in the atmosphere of the earth, by P. Duckert; pp. 236-290, 16 figs. (6) New methods for the determination of the figure of the earth, by F. Hopfner; pp. 291-372, 9 figs. (7) On the dynamics of the forms of movement on the surface of the earth, by F. M. Exner; pp. 373-445, 65 figs. (8) Index of authors, pp. 446-448.

(640) Gutenberg, Prof. Dr. B. Handbuch der Geophysik (Handbook of Geophysics). Verlag von Gebrüder Borntrager in Berlin, W. 35. This book will be issued in 10 volumes. The contents of the volumes are as follows: Vol. 1, The earth as a planet (in press). Vol. 2, Structure of the earth: No. 1, Cooling and temperature of the earth, by B. Gutenberg; Chemistry of the earth, by Berg; Age of the earth, by A Born; physical structure of the earth by B. Gutenberg; 564 pages, 183 figs. 1931. Price of the single

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(643) Tinsley, H., and Co. Geophysical prospecting by electrical methods. Telegraph and Scientific Instrument manufacturers. Werndee Hall, South Norwood, London, S.E. List No. 62, August, 1931. Some of the apparatus (ratiometer, D.C. potentiometer) used for geophysical prospecting by electrical methods is described.

10. PATENTS

(644) VERFAHREN ZUR SEISMISCHEN BODENFORSCHUNG

(METHOD OF SEISMIC EXPLORATION)

Dr. Richard Ambronn of Göttingen

German Patent 521,573. Patent issued March 24, 1931.

This patent discloses a method of seismic exploration by means of elastic waves artificially produced by explosive charges. The charges, which are ignited simultaneously, are arranged in one plane in squares and in such a way that a preferable direction of the effect is obtained. The receiving instruments are placed inside of the cone of the waves.

Claims allowed - 5.

(645) PROCÉDÉ DE PROSPECTION DE LA CRÔUTE TERRESTRE

(METHOD OF PROSPECTING THE EARTH'S CRUST)

Heinrich Löwy of Austria

French patent 657,367. Patent issued May 22, 1929.

The present invention concerns the prospecting of the earth's crust and is characterized by measuring at least two electrical values, one of which (the proper length of the wave of an electrical oscillator, angle of reflection, position of an extreme value of interference) depends, in case of the presence of a conductive layer, only on the distance from this layer, and the other (damping or amplitude of the vibrations of the oscillator) of which depends besides on the nature of the material.

Claims allowed - 5.

(646) MÉTHODE ET APPAREIL SERVANT A MODIFIER LA DISPOSITION DES RADIATIONS D'UN CORPS ET PLUS PARTICULIÈREMENT SON APPLICATION À LA RECHERCHE ET À LA PROSPECTION DU SOUS-SOL

(METHOD AND APPARATUS SERVING FOR MODIFYING THE DISPOSITION OF RADIATIONS OF A BODY, IN PARTICULAR ITS APPLICATION FOR INVESTIGATING AND PROSPECTING OF THE SUBSOIL)

(Invention made by Hugues-Emile-Joseph Roche and Leon-Marie-Antoine Bidreman)

Société Anonyme Robivir of France (Rhône).

French patent 668,808. Patent issued November 7, 1929.

This invention concerns the method and appáratus by which modifications in the position of the negative electrons and of the free positive ions of a body, and possibly also changes in their velocities and their paths, may be created. Therefore the invention makes it possible to modify the radiations emitted by this body by exposing it to an electrostatic field.

Claims allowed - 2.

(647) PROCÉDÉ ET DISPOSITIF DE RECHERCHES GÉOPHYSIQUES PERMETTANT DE RECONNAÎTRE ET D'EXPLORER LE SOUS-SOL

(METHOD AND APPARATUS OF GEOPHYSICAL PROSPECTING BY WHICH IT IS POSSIBLE TO INVESTIGATE AND EXPLORE THE SUBSOIL)

Mines domaniales de Potasses d'Alsace of France (Haut-Rhin)

French patent 685,487. Patent issued July 10, 1930.

The method of geophysical prospecting disclosed in this patent consists mainly in measuring the earth resistance between two electrodes. The resistance of the earth is calculated from the maximum readings of the voltmeter.

Claims allowed - 3.

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3 The first figure refers to the number of the abstract, the second to the method of prospecting as indicated in the Table of Contents, and third to the page.